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PREFACE

In this volume will be found papers on mining, geology, mine ventilation, mining methods, ground movement and subsidence, coal and coke, and on petroleum. It is a companion to Volume 73 already issued and containing papers on non-metallic minerals, milling and concentration, non-ferrous metallurgy, iron and steel and those presented before the Institute of Metals Division. Together the two present the major results of the work of the Institute as reflected in the meetings at Casper, Wyoming, and Salt Lake City in the fall of 1925 and at New York and Pittsburgh in 1926. Few of the papers read before the Petroleum Division have been included for the reason that they have received separate publication in the volume on "Petroleum Development and Technology in 1925" distributed in the spring of 1926. Others read at the Division meeting at Tulsa in October last are being held for inclusion in a similar volume to be published following the next annual meeting. Similarly papers read at the Detroit meeting of the Institute of Metals Division are being held to be included in a special volume to be distributed in the spring.

In the course of the year 134 papers have been printed in pamphlet form and sent to members interested and to libraries. A considerable additional number have been printed in full in Mining and Metallurgy and sent to all members and all pamphlets printed have been noted in abstract in that journal. In a few instances papers printed as pamphlets or in Mining and Metallurgy are not reprinted in either the Transactions or in the special volumes, they having received in the judgment of the Papers and Publications Committee adequate circulation in their first form of publication. In the main such papers represent essentially progress reports of investigations, summary statements of the present state of the art, or summaries or descriptions of local plants or deposits of especial interest in connection with a particular meeting. In view of the heavy expense of printing, the Committee finds it necessary to avoid reprinting and duplication as far as possible. Any paper, however, printed by the Institute is available to any member and all papers are furnished to the leading technical libraries.

With further development of the present policy of giving prompt publication and distribution of papers submitted as pamphlets or in Mining and Metallurgy and the assembling into special volumes of the papers in the field of each of the two Divisions of the Institute, it is expected that a single volume of Transactions will be sufficient to include the papers of

each year. As the amount of material available becomes sufficient, it may well happen that special volumes of the pamphlets in classes other than those covered by the two Divisions may be issued.

H. Foster Bain.
Secretary

New York, N. Y.
December 15, 1926.

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* The purpose of this division is to obtain papers of high merit for publication by the Institute on production, transportation, refining and utilization of petroleum and its products.

PROCEEDINGS

Pittsburgh Meeting

The 134th general meeting* of the Institute was held at Pittsburgh, Pa., Oct. 5 to 9. The attendance totaled approximately 600 members and guests.

The formal opening session was held in the Music Hall of Carnegie Institute, Oct. 5, Graham Bright presiding. President Samuel A. Taylor welcomed the visiting members and guests and reviewed briefly the advancement of mining in the past half century. John S. Unger, of the Carnegie Steel Co., spoke of the Institute's previous meetings in Pittsburgh (1872, 1879, 1886, 1896 and 1914) and welcomed its return.

The open-hearth group held their 4th semi-annual conference at the William Penn Hotel, Oct. 5 and 6. A round table conference on the combustibility of coke was held Oct. 5, at the U. S. Bureau of Mines. Sessions of the Coal and Coke Committee were held Oct. 6 and 8, for the reading of papers, J. P. Williams and M. D. Cooper presiding and about 100 members attending. S. L. Goodale and James Aston were the respective chairmen of the sessions on iron and steel held in Carnegie Institute on Oct. 6 and 8. Fifty members attended these sessions. A session on ventilation was held on Oct. 8, in the auditorium of the Pittsburgh Experiment Station of the U. S. Bureau of Mines, with George S. Rice in the chair and 55 in attendance. Natural gas problems were considered at a session held Oct. 8, at the Mellon Institute of the University of Pittsburgh, and presided over by Prof. Roswell Johnson.

The entire day of Oct. 7, was given over to a boat trip on the Monongahela River. The party left at 9 a.m. from the Allegheny River side and passed around the point of intersection of the two rivers to Clairton, 21 miles up the Monongahela, where at the Carnegie Steel Co.'s great by-product coking plant, established in 1916, opportunity was given the members to see the filling of an oven with coal, discharging an oven, quenching the hot coke, and the recovery of ammonium sulfate.

Returning down the river the party passed the large U. S. Glass Co. works at Glassport, and stopped at the plant of the National Tube Co. at McKeesport, where the sequence of operations from making pig iron in the blast furnace, converting it to steel in the Bessemer converter, casting into ingots, reheating and rolling into strips of the proper size and forming and welding into tubes were interestingly shown. Many other great plants were passed en route to Pittsburgh.

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* For news story of meeting see Mining and Metallurgy (Nov., 1926) 470.

On the afternoon of Oct. 8, all groups visited the experimental mine and explosives experiment station of the U. S. Bureau of Mines at Bruceton, 9 miles from Pittsburgh, where apparatus for determining the composition of mine air was exhibited. The members saw the demonstration of ignition of clouds of coal dust by an electric arc and by an open-flame lamp; the operation of rock-dust barriers; ignition of a gas-air mixture in a 12-in. tube; the effect of a blown-out shot of a permissible explosive in the presence of coal dust contrasted with a blown-out shot of black powder in the presence of coal dust, and the ignition of a keg of black blasting powder by an electric current.

A spectacular event was the explosion in the experimental mine of 800 lb. of fine Pittsburgh coal dust distributed in the first 400 ft. of the main entry. This was ignited 180 ft. from the pit mouth by a blown-out shot of black powder. The entry in the coal dust zone was rock dusted by the distributor and the effect of the rock dust in preventing the flame traveling into the mine was observed.

The meeting was concluded by a reception by President and Mrs. Taylor, and a banquet, both held at the Hotel Schenley. Howard N. Eavenson was the toastmaster on the latter occasion. The J. E. Johnson, Jr., prize for 1926 was presented to S. P. Kinney by past-president J. V. W. Reynders. President Taylor and C. F. Kettering, vice-president of the General Motors Co., were speakers.

Technical Sessions

Open-hearth Conference

E. A. Whitworth, Chairman

Manufacture of Forging Steel by the Basic Open-hearth Process. R. L. Cain
The Basic Open-hearth Charge. Paul H. Shaeffer
Mineralogical Composition of Hearth Bottoms. W. J. McCaughey
Effect of the Refractories on Steel Making. G. M. Demorest
Absorption of Sulfur During Melting in the Open-hearth Furnace. C. H. Herty, Jr.
Desulfurizing Action of Manganese in Iron. C. H. Herty, Jr., and J. M. Gaines, Jr.

Coal and Coke

J. P. Williams, Chairman

The Pittsburgh Coal Bed of Pennsylvania,* G. H. Ashley
The Pittsburgh Coal Bed of Ohio.* J. A. Bownocker
The Pittsburgh Coal Bed of West Virginia.* I. C. White.
The Coal Mining Plant of the Buckeye Coal Co. at Nemacolin, Pa.
(1) Resume of the Construction Work. A. W. Hesse
(2) Mine Operation. W. Z. Price

M. D. Cooper, Chairman

Mining Methods in the Pittsburgh District.† N. G. Alford and B. F. Hoffacker
X-ray Studies of Coal and Coke. Ancel St. John

* These papers are printed in this volume as one paper under the title The Pittsburgh Coal Bed.

† Submitted by the Pittsburgh District Sub-committee on Coal and Coke.

Time Element in the Control of Face Conditions in Coal Mining. H. F. McCullough
Appraisal of Coal-property Values. H. M. Chance

Iron and Steel

S. L. Goodale, Chairman

Trend of Development in the Wrought Iron Industry. James Aston
The Largest Steam-hydraulic Forging-press. W. J. Priestley
Bend Tests of Galvanized Sheet Steel. H. A. Stacy
Progress Report on the Effect of the Open-hearth Process on Refractories. F. W. Schroeder and B. M. Larsen
Optical Temperature Measurements in Open-hearth Furnace. B. M. Larsen and J. W. Campbell

James Aston, Chairman

Specific Efficiency of the Blast Furnace. Richard Franchot
Iron and Steel Scrap in Blast-furnace Burdens. W. C. McKee
Composition of Iron Blast-Furnace Slags. Richard S. McCaffery, Joseph F. Oesterle and Leo Schapiro

Ventilation

George S. Rice, Chairman

Occurrence of Fire Damp in Bituminous Mines. Frank Haas
State Coal Mining Laws Concerning Ventilation. John A. Garcia
Mine-air Flow. G. E. McElroy
Application of Kutter's Formula to Gases. F. Ernest Brackett

Gas

R. H. Johnson, Chairman

The Natural Gas Industry. S. W. Meals
Estimation of Natural Gas Reserves. Ralph E. Davis
Relation of By-product Coke Ovens to the Natural Gas Supply of the Pittsburgh District. Harold J. Rose

Petroleum Development and Technology in 1925*

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PART I

PART II—PRODUCTION ENGINEERING

* Issued as Pamphlet No. 1570-G with MINING AND METALLURGY, April, 1926. Contains papers presented at symposiums of the Petroleum Division, at the New York meeting, Feb. 15–17, 1926; the Los Angeles, Cal., joint session, Jan. 21, 1926; and the Casper, Wyo., meeting, Aug. 28–30, 1925.

Part III—Refining Technology

PART IV—TRANSPORTATION ENGINEERING

PART V—PRODUCTION

PART VI—PETROLEUM ECONOMICS

PAPERS

Electrical and Electromagnetic Prospecting

By Hans Lundberg,* New York, N. Y.

(New York Meeting, February, 1926)

The art of prospecting, like other branches of mining, has developed rapidly during the last ten years, mainly because of the advancement of mining geology, but also because of the development of "geophysical methods," which are based on the different physical characters of minerals and rocks, *e. g.*, specific gravity, elasticity, and magnetic and electric properties. These methods have proved of great value, especially in countries where the rock is covered by soil and direct geologic investigation is out of the question.

In prospecting for orebodies, the methods based on studying differences in electrical conductivity have proved most successful; for which reason, the methods of electrical prospecting have been subject to the most energetic and systematic work, especially in Sweden. This paper gives a brief description of the methods of electrical prospecting now in use, especially the new electromagnetic methods, which have not been described before.

The great differences in the electrical properties, especially in conductivity, between ore minerals and rock minerals are fundamental to all methods of electrical prospecting; for which reason, the electrical conductivity of ore minerals and rock minerals has been carefully studied in connection with and previous to the development of the various methods of prospecting with electricity.

Ore minerals with a metallic luster, such as chalcopyrite, iron pyrite, galena, specular iron, magnetite, etc., also graphite and a number of coals, have considerably higher conductivity than the surrounding rock and soil (see Table 1). The figures of the table denote the specific resistance W, expressed in ohms for a cube with an edge of 1 centimeter.

To obtain an idea of the applicability of the various methods of electrical prospecting, it is necessary to investigate the elements controlling the results obtained by any certain method. The best way of doing this is by adopting simple, idealized cases, so that conditions may be mathematically expressed. Then, the application of these expressions may be tested on more complicated cases by work in the laboratory. Finally, results obtained in practice are compared to tests in the laboratory and thus the general application of the theory is tested.

* Field manager, Swedish American Prospecting Corp.

TABLE 1

Minerals	Ohms per Cm³	Rocks and Ores	Approximately Ohms per Cm³
Calcite	5.0×10^{14}	Quartzite, limestone, sandstone granite	10^{11}
Quartz	3.8×10^{11}	Leptite, schists	10^9
Mica	1.5×10^{10}	Greenstone	10^8
Serpentine	2.0×10^4	Hematite and spathic iron ore	10^8
Siderite	7.1×10^3	Zinc-blende ore (non-ferrous)	10^8
Marcasite	10.0	Specular iron ore	10–100
Chalcopyrite	1.0	Magnetite ore	10–100
Molybdenite	0.8	Schists with pyrrhotite	5–100
Magnetite	0.6	Galena ore	1
Specular iron	0.8 – 0.4	Chalcopyrite ore	0.1
Graphite	0.03	Sulfur pyrite ore	0.1
Pyrite	0.02		
Pyrrhotite	0.01		
Galena	0.003		

The Methods

The methods of electrical prospecting may be divided into two principal groups, "potential" and "electromagnetic." The potential methods, which are the older, are based on the investigation of the potential in an electrical field by tracing equipotential curves and measuring the differences in potential between different points. The electromagnetic methods are of more recent origin. By them, the direction and intensity of an electromagnetic field are investigated.

The Potential Methods

The potential methods were tried as early as about 1830 and came into greater practical use in 1918, in conjunction with the introduction of linear electrodes. As the electrical field obtained by means of linear electrodes is simple and homogeneous, the equipotential curves become straight lines and the potential V_p at any point can be calculated in accordance with the equation:

$$V = \log_e \frac{(r_1 + l_1)\,(r_2 + l_2)}{x^2} \times \frac{y^2}{(r_3 + l_3)\,(r_4 + l_4)};$$

where the letters refer to Fig. 1.

The strength of the disturbances an orebody causes will be magnified and the interpretation simplified in a field obtained by linear electrodes. The application and some of the practical results that have proved the success of this method have already been published.

The potential methods have been used, since 1918, at about seventy ore deposits in Sweden, Norway, Finland, Spain, Italy, the United States and Canada, and resulted in the discovery of a number of ore-

bodies previously unknown. Some of these proved to be of great commercial value. The area investigated exceeds 40,000 acres. The

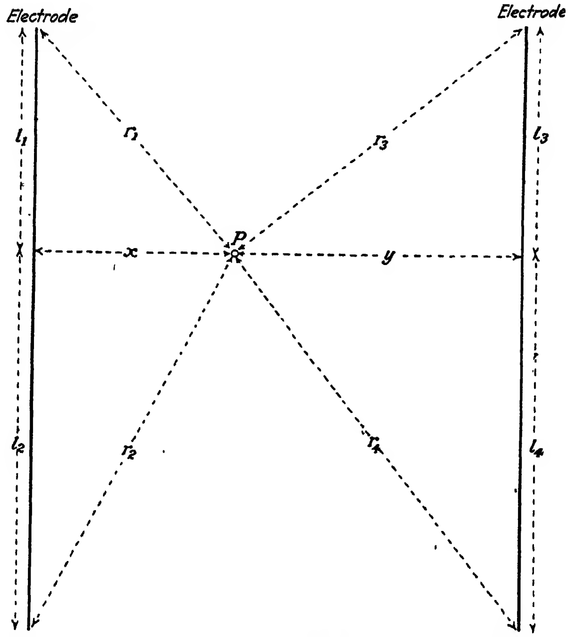

Fig. 1.—Designation for equation of potential.

largest and, so far as execution is concerned, most rational electric investigation has been made in the Skelleftea region, in Northern Sweden,

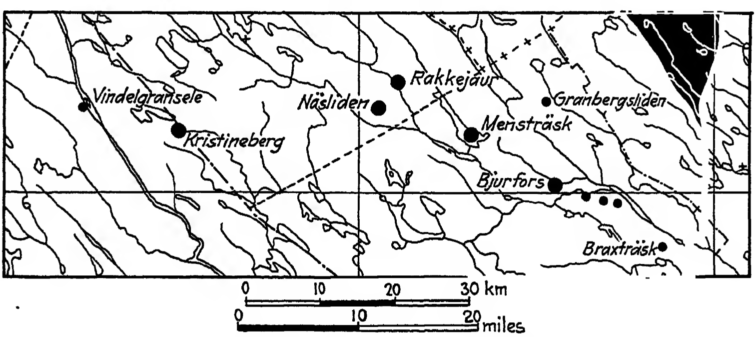

Fig. 2.—Central part of Skelleftea district, with deposits located up to present time.

where 20,000 acres have been investigated. The scope of this investigation proves that a complete electrical investigation of an entire ore

district is already possible from a technical, as well as an economic, point of view. The orebodies, chalcopyrite, pyrite, auriferous arsenopyrite and zinc ore, found by potential methods, are divided into two large deposits, Kristineberg and Bjurfors, and several minor ones, with a total area of about 100,000 sq. ft. (see Fig. 2).

Electromagnetic Methods

The electromagnetic methods for prospecting were first adopted about 1907; the main development occurred in 1921, when Karl Sundberg, a Swedish mining engineer, who has prepared the following description, began to experiment with a number of these methods. They have been used on a large scale and with good results, since 1922, in prospecting in Sweden and Norway. They are now being exploited, also, in the United States and elsewhere.

In the electromagnetic methods, current is caused to flow in an orebody and the electromagnetic disturbances caused by it are examined. The methods, according to the manner of causing the current to flow, may be divided into:

1. Methods in which current is caused in the orebody inductively; *i.e.*, by means of closed loops of cables insulated from the ground.
2. Methods in which current is caused galvanically; *i. e.*, by means of wires connected with the ground. In this case, the current is likewise caused inductively by the cables to the points of contact with the earth.
3. Methods in which current is transmitted capacitively; *i. e.*, by means of open wires insulated from the ground, antennæ.

These methods also include "wireless" or "radio" methods.

The disturbances caused by the orebody can be investigated:

1. By direct reading, in which case the strength of the electromagnetic field is read off at every point of observation.
2. By comparative readings, in which case the strength of the electromagnetic field, at a certain point, is compared with that of the field at another point.

The methods that Sundberg has elaborated involve different combinations of the principles that form the basis of the groupings given above. According to the methods in question, current is thus raised in the orebody, either inductively, galvanically or capacitively, or by a combination of these procedures, and the electromagnetic field is investigated either directly or comparatively. Applications have been made for patents for the inventions in question.

Theory

In the case of an inductive supply of current, the circuit on the ground (the primary circuit) forms, together with the orebody, a short-circuit transformer, the secondary circuit of which is the orebody. In order to

indicate the position and outline of the orebody, an examination is made of the course of the electromagnetic field generated by the current in the orebody (the secondary field). To begin with, it is necessary, therefore, to comprehend which factors determine the course of the secondary field. Let AB, Fig. 3, be the primary circuit, CD the secondary circuit (the ore-

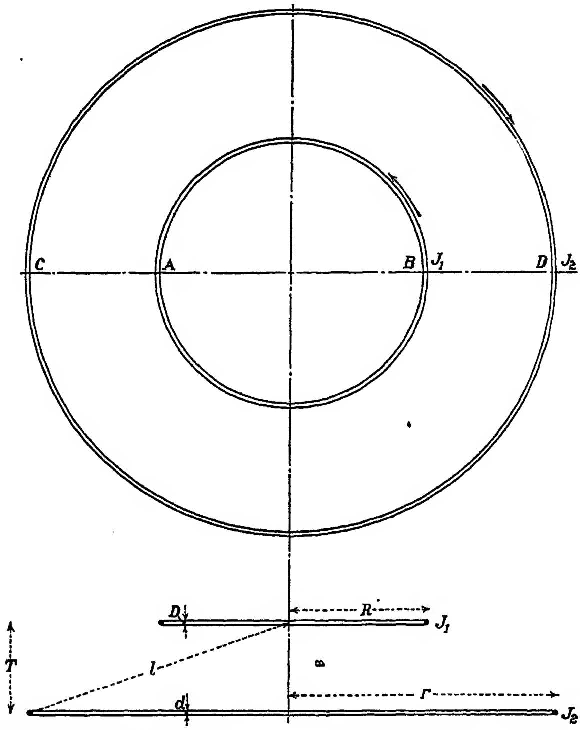

FIG. 3.—DESIGNATION FOR EQUATION OF ELECTROMAGNETIC FIELD INDUCTIVELY PRODUCED.

body), and suppose that both are circular. It can then be proved that the following equation is obtained:

$$V^A{}_S = \frac{Rr^2 \cdot 2\pi v \cdot M}{l^3 \cdot \sqrt{\dfrac{4k^2r^2}{d^4} + 4\pi^2v^2L^2}} \cdot V^A{}_P,$$

where $V^A{}_P$ = intensity of primary field at point A;
$V^A{}_S$ = intensity of secondary field at point A;
R, r, l, d = distances shown;
k = specific resistance of ore circuit;
v = frequency;
M = mutual inductance between primary and secondary circuits;
L = inductance of secondary circuit.

The secondary field thus depends on the dimensions and mutual positions of the primary circuit and the orebody and on the frequency, as well as the resistance of the orebody k and permeability μ (as M and L

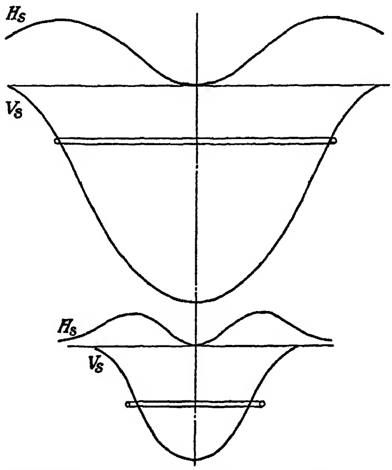

FIG. 4.—SECONDARY ELECTROMAGNETIC FIELDS, SHOWING POSITION AND DIAMETER OF ORE CIRCUIT.

are functions of μ). The secondary field becomes larger with falling resistance in the secondary circuit (the orebody) and the resistance can be determined by measuring V_S at different frequencies.

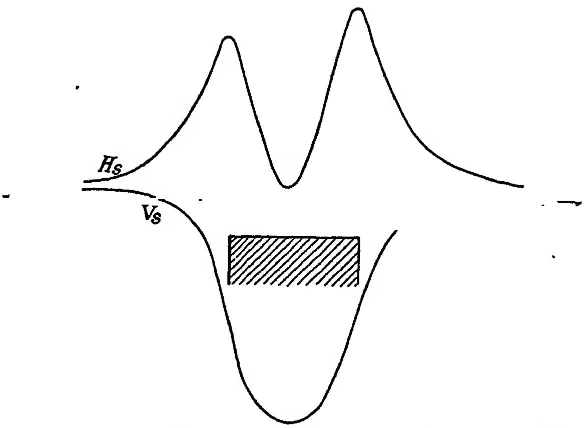

FIG. 5.—SECONDARY ELECTROMAGNETIC FIELD, ACCORDING TO LABORATORY TESTS, ON A SMALL SCALE.

The secondary field at an arbitrary point has a vertical component, V_S, as well as a horizontal component H_S. Fig. 4 shows the course of

V_S and H_S in this case, as well as a case when the diameter of the "ore circuit" is doubled. It ought to be quite evident that the position and diameter of the circuit can be indicated by the secondary field.

As analogous secondary fields will be obtained for any outline of the orebody, the position and form of the orebody can, in the case of the

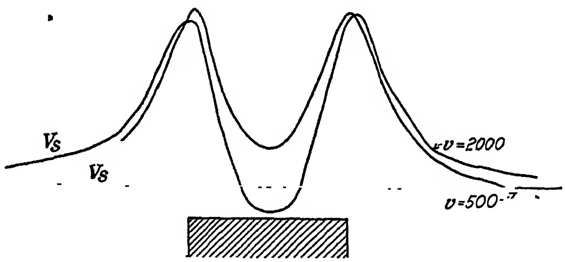

FIG. 6.—SECONDARY ELECTROMAGNETIC FIELD AT TWO FREQUENCIES, ACCORDING TO LABORATORY TESTS ON A SMALL SCALE.

inductive method, be determined from the secondary field. In addition, at least in favorable cases, the strength of the secondary field gives an idea of the electrical conductivity and permeability of the ore. Fig. 5 shows the secondary field according to laboratory tests carried out on a small scale; Fig. 6, the secondary field at two different frequencies according to

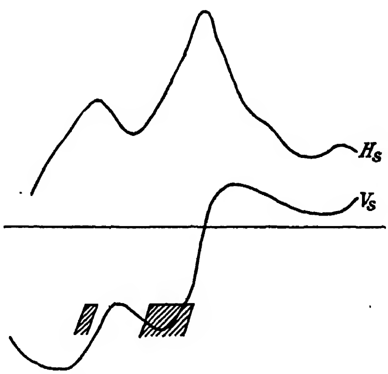

FIG. 7.—SECONDARY ELECTROMAGNETIC FIELD ABOVE TWO PARALLEL OREBODIES, WHICH WERE DISCOVERED BY ELECTROMAGNETIC INVESTIGATION.

similar tests; Fig. 7, the secondary field above two parallel orebodies, which were discovered by electromagnetic investigation.

With the galvanic method are used, for example, two linear electrodes, E_1 and E_2, according to Fig. 8. If A is a vertical lamellar orebody

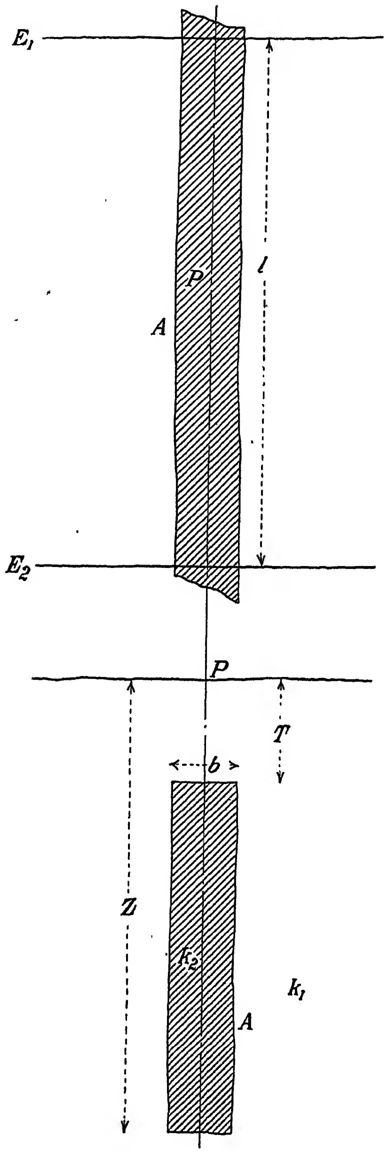

FIG. 8.—DESIGNATION FOR EQUATION OF ELECTROMAGNETIC FIELD GALVANICALLY PRODUCED.

located (beneath the earth) at the depth T, the following equation approximately applies:

$$H^P{}_S = \frac{2b(k_1 - k_2)}{\pi^2 \cdot l \cdot k_2} \cdot \log_e \frac{Z\sqrt{l^2 + T^2}}{T\sqrt{l^2+Z^2}} \cdot H_P;$$

where $H^P{}_S$ = horizontal component (at point P) of secondary field generated as a result of current in ore;

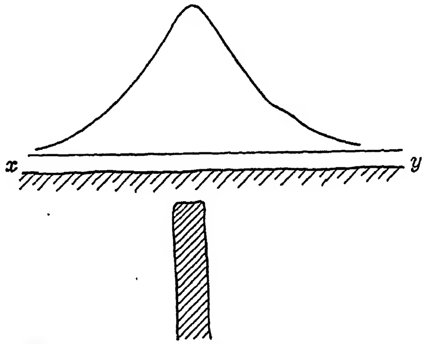

FIG. 9.—VALUE OF H CALCULATED ABOVE CONDUCTING BODY.

H_P = strength of electromagnetic field at P, if there were no orebody;

l, b, Z, T = distances shown in figure;

k_1 = specific resistance of surrounding rock;

k_2 = specific resistance of ore.

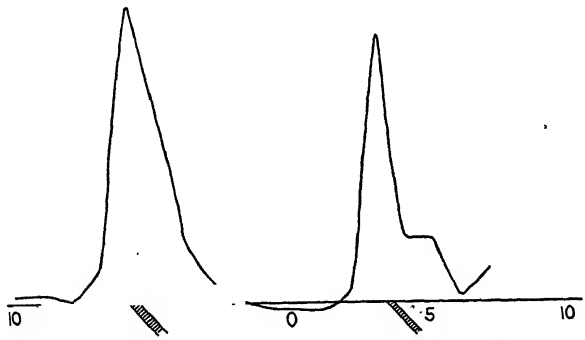

FIG. 10.—VALUE OF H SHOWS TWO PARALLEL OREBODIES ACCORDING TO INVESTIGATIONS CARRIED OUT IN FIELD.

The quotient between the specific resistance of the rock and the orebody thus determines, apart from the geometrical dimensions, the strength of the secondary field.

If the value of H is calculated at different points along the line xy, for $l = 500$, $b = T = 10$, $Z = 100$, $k = 1000$, the values indicated by Fig. 9 are obtained, *i. e.*, a pronounced maximum is obtained above the

orebody. The position of the latter is thereby determined. Fig. 10 denotes the value of H above two parallel orebodies according to investigations carried out in the field.

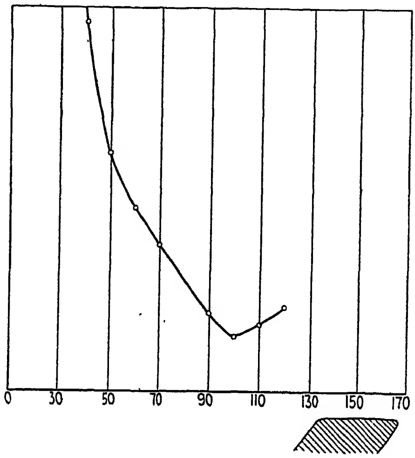

FIG. 11.—OREBODY NOTICED BY MEASURING CURRENT IN RECEIVING ANTENNA.

As regards the theory of the capacitive methods, especially the "wireless" or "radio" methods, it should be pointed out that the earth, even where only slightly damp, has so strong a conductivity that the short electromagnetic waves used in these methods are rapidly

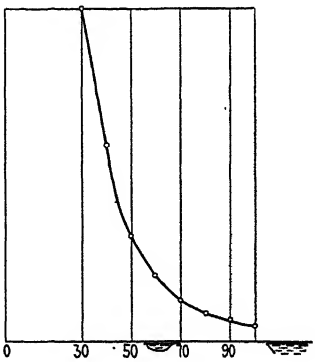

FIG. 12. FIG. 13.

FIG. 12.—OREBODY NOTICED BY MEASURING CURRENT IN RECEIVING ANTENNA.
FIG. 13.—WATER SHOWS NO INFLUENCE ON CURRENT IN RECEIVING ANTENNA.

absorbed. Thus, at even shallow depths, the energy is only a small fraction of what it is on the surface. The principle involved is shown by the equation calculated by J. Zenneck[1] for the transmission of electromagnetic waves in different media.

[1] *Annalen d. Physik* (1907) **23.**

With this disadvantage, it is inconceivable that the position of orebodies at any considerable depth can ever be determined by electromagnetic waves except, possibly, in extremely dry countries. In wet countries, the absorption would be too great to indicate deep ore, although where the ore is shallow some success has been obtained by Karl Sundberg in Central Sweden, using a method devised by him. By this method a movable receiving antenna is used, in which the strength of the current I is observed. The strength of this current depends, not only on the distance from the transmitting station, but on the specific conductivity and dielectric constant of the subjacent ground. As these factors for ore minerals are very different from those of ordinary rocks, the current I in the receiving antenna will be different when the antenna is in the neighborhood of an orebody from what it is on barren ground.

In a total of twenty-two investigations, distinct indications were obtained in seventeen cases, indistinct in three, and none in two. As shown by Figs. 11 and 12, taken from these operations, the ores make their presence known by the current in the receiving antenna increasing in front of, and decreasing behind, the orebody. Fig. 13 shows that, by these investigations, water did not cause indications, probably because the electric properties of the water in these cases are scarcely discernible from those of the soil. In comparison with other electromagnetic methods, the "wireless" methods are at present of no importance and will not again be referred to.

Summary

By supplying the current inductively or galvanically into an orebody and determining the secondary field generated by the current in the orebody, the position of the latter may be fixed; and, in the case of the inductive method, the physical nature of the ore may be determined.

Practice

Preceding Laboratory Tests

When searching for orebodies, it must first be ascertained whether the nature of the ore minerals is likely to be such as to make the investigation successful. From what has been said, the decisive factors are the specific conductivity, the permeability and inductance of the ore and of the surrounding rocks, *i. e.*, the electric and magnetic properties of the ore and the rock, which properties, therefore, should be examined. In addition to this, the geological conditions, such as stratification, the sequence of same, the depth of the ore-bearing horizon, the extension of the orebodies, etc. determine if the electrical investigation is at all possible. Further, the method of investigation that would be most suitable must be ascertained. In doubtful cases, a laboratory investigation can be made on a small scale under conditions analogous to those expected in the field.

The field work should, therefore, be preceded by work in the laboratory, as this is by no means insignificant, such as tests of the physical properties of the ore and rocks, as well as examination of model orebodies. Fig. 14 illustrates the S. A. P. C. laboratory in Stockholm, where such tests are made.

Field Work

The investigations are made in order to recognize and analyze the secondary field caused by the orebodies and other conductors. The observations are generally made at points marked on lines staked out on

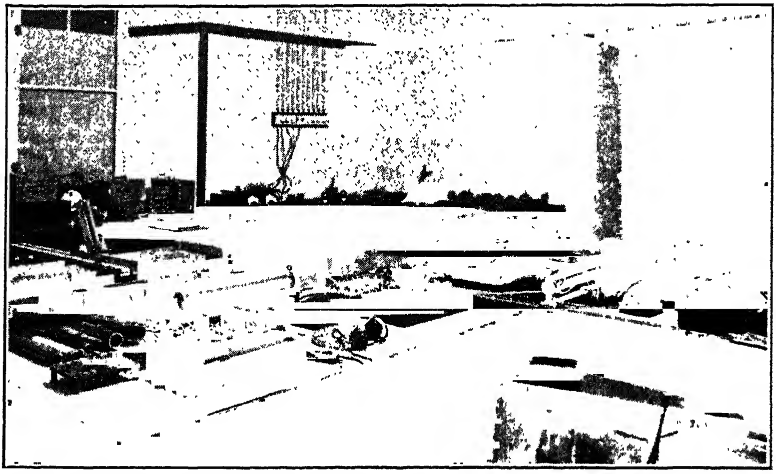

FIG. 14.—SWEDISH AMERICAN PROSPECTING CORPORATION'S LABORATORY IN STOCKHOLM.

the ground, but they may be made at arbitrary points, the positions of which must then be specially surveyed. As a rule, a preliminary investigation is first made along lines 150 to 250 ft. apart at right angles to the general direction of the strike. In doing this, the course of the vertical component of the secondary field is investigated. This investigation indicates the approximate position of conducting bodies. A prepared area of 250 acres is, as a rule, investigated in about three days by two men, in which case five unskilled hands are required. The indications resulting from the preliminary survey are then examined in detail, and in doing so, the observations are made along lines 30 to 75 ft. apart and the points of observation are more frequent than in the reconnaissance. In the detailed investigation the horizontal, as well as the vertical, component is read off. In cases of deep-dipping strata, one set-up is, as a rule, required in order to distinguish the position of the hanging wall of the conducting body and another set-up to distinguish the foot wall. The results of these detailed investigations are then put together on a map,

the outlines of the conductor being obtained by connecting the various points of the boundary lines. As a rule, it has been possible to determine the outlines of the conductors to within a few feet, even when the over-

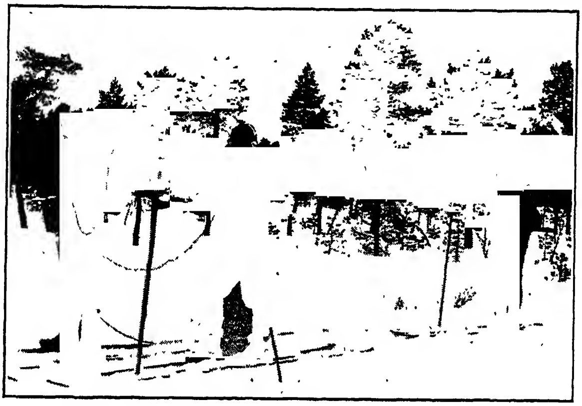

FIG. 15.—ELECTROMAGNETIC FIELD WORK.

burden has exceeded 60 ft. The method has not yet been tested in the case of deeper orebodies, but there is no reason to suppose that the results will deteriorate to any appreciable extent for greater depths, if conditions in other respects are favorable.

FIG. 16.—ELECTROMAGNETIC FIELD WORK.

Figs. 15 and 16 show field investigations. Both in 1923 and 1924, surveys have been carried out in the north of Sweden throughout entire winters, at times at a temperature of 40° below zero (Fahrenheit).

Results

The electromagnetic methods have been in practical operation since the summer of 1922. Since then there has been investigated, within the Skelleftea district alone, an area of 30,000 acres, besides considerable areas in other parts of Sweden and Norway.

Geology of the Skelleftea District

The Skelleftea district is a pre-Cambrian formation of schistose volcanics and sediments surrounded by younger granites, corresponding to Keewatin in Canada and the United States. Systematic electric and geological investigations have shown that this field contains a large number of deposits of pyrite, chalcopyrite, auriferous arsenopyrites, and zinc ores. These usually comprise lenticular orebodies 100 to 1000 ft. in length and varying from 10 to over 150 ft. in width. The entire region is covered by deep moraine, sand, swamps and lakes, which greatly increased the difficulties of discovering the ore deposits.

The extent of the region and the long winter (seven months of snow) have exacted methods that work rapidly, cheaply, and under the severest climatic conditions. Great technical difficulties were also experienced, owing to pyritic and graphitic schists with good electrical conductivity.

In the district mentioned, four new ore fields have been discovered by means of electromagnetic investigations, namely, Naesliden, Rakkejaur, Menstraesk, and Braxtraesk (see Fig. 2). These fields contain extensive pyrite, chalcopyrite and arsenopyrite ores, with a total area of 250,000 to 300,000 sq. ft.

Rakkejaur Ore Field

In the fall of 1921, a large number of ore boulders were discovered at Rakkejaur farm. In 1922, a careful electrical investigation was made of the field, followed by excavating operations and diamond drilling; during 1923 and 1924, the operations were continued. The rocks in this field consist of gray, schistose quartzitic porphyries, with occasional beds of tuffitic breccias and graphitic black schists, disseminated with pyrrhotite with good electrical conductivity. The orebodies lie in the contact between the schistose porphyry and the black schist and occur in a relatively poor breccia of pyrite, pyrrhotite, chalcopyrite, boulangerite, and arsenopyrite, in which rich sulfur, copper, and arsenic orebodies (40 per cent. sulfur, 2 – 3 per cent. copper, 20 – 25 per cent. arsenic, 0.5 oz. gold) are to be found. The soil varies from a few feet up to 30 ft. Fig. 17 shows the results of the exploration work and the electrical investigations. As the difference in conductivity between the orebodies and the black schist is relatively slight, it is natural that difficulties were experienced in distinguishing the boundaries of the orebodies towards the

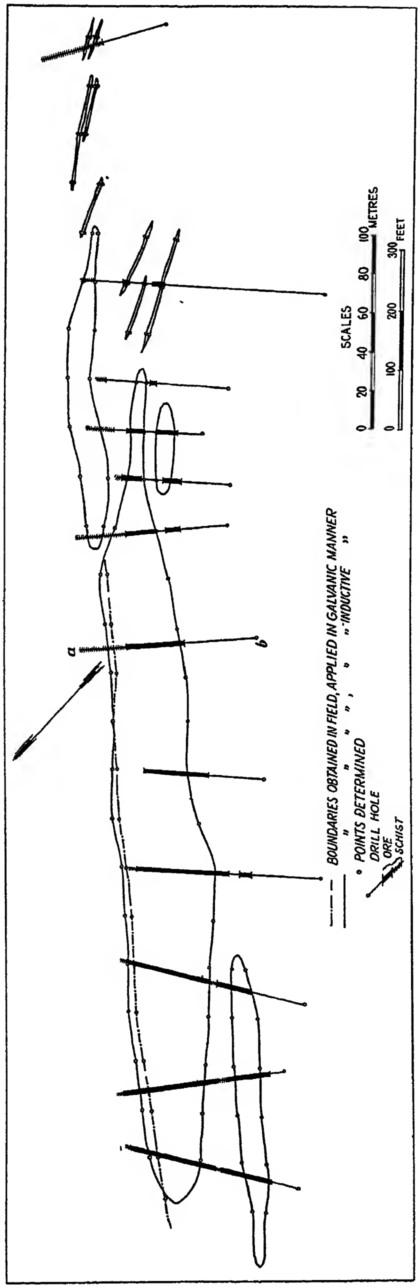

FIG. 17.—RESULTS OF EXPLORATION WORKINGS AND ELECTRICAL INVESTIGATIONS AT RAKKEJAUR ORE FIELD.

black schist. It will be seen from the map, however, that the said boundaries were obtained satisfactorily, both in distributing the current inductively and galvanically into the orebodies. Fig. 18 illustrates the secondary fields obtained by investigations along line *ab*. The boundary points of the orebody obtained are the result of this electrical investigation.

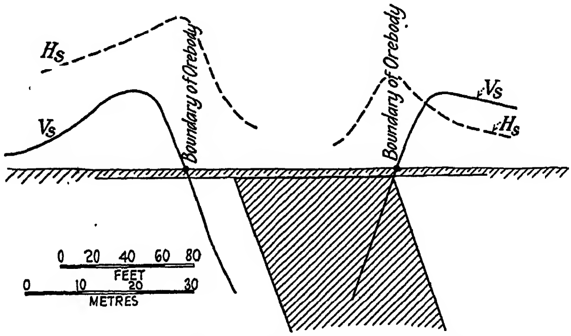

FIG. 18.—SECONDARY ELECTROMAGNETIC FIELD ALONG LINE *ab* AT RAKKEJAUR ORE FIELD.

Braxtraesk

In 1921 to 1923, several boulders of pyrite and chalcopyrite were found at Braxtraesk in a sericite-quartzite country. The suspected areas were, therefore, investigated. Good electrical indications were then obtained. Two holes were drilled through the indications in 1924, when ore was found in both the holes. The overburden is about 40 ft. Fig. 19 depicts the result of the electrical reconnaissance and the detailed investigations, as well as drilling; Fig. 20 the secondary field along the line *ab*.

Menstraesk

In the summer of 1921, several boulders of ore were found in the Menstraesk region; in 1922 and 1923, a great number of new boulders were discovered. Fig. 21 is a map of the Menstraesk area, showing the position of the ore boulders. Because the ore boulders are spread over a large area and there are hardly any exposures of the rock, it has been difficult to define geologically the area within which the ores are probably to be found. A relatively large area, about 10,000 acres, therefore, had to be investigated electrically. As shown by Fig. 21, innumerable electrical indications were obtained with a total length of not less than 15 miles, among which only 1500 ft. contain commercial areas. The majority of the indications originate from disseminated graphitic schists, most of which have an extremely good conductivity. On this account, detailed and qualitative electrical investigations have caused a considerable amount of work. The number of electrical observations carried out in this field thus amount to approximately 150,000.

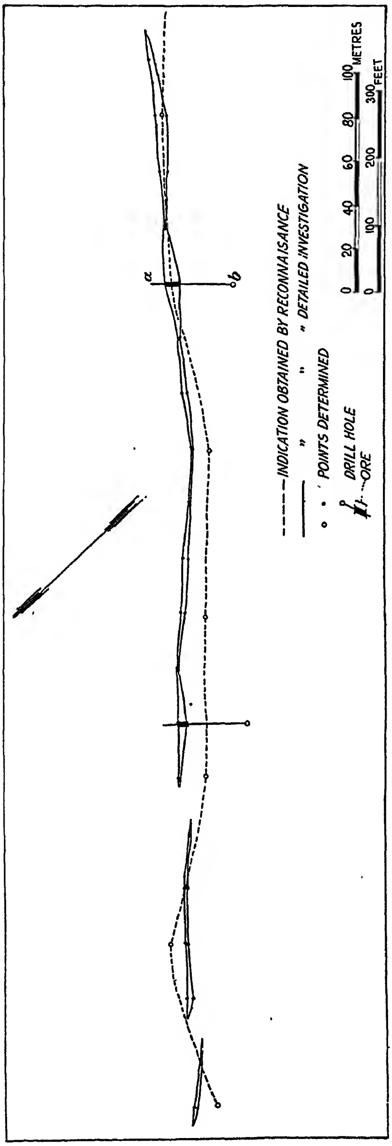

FIG. 19.—RESULTS OF DRILLINGS AND ELECTRICAL INVESTIGATIONS AT BRAKTRAESK ORE FIELD.

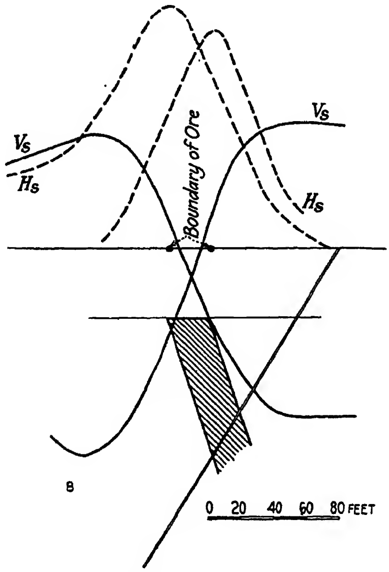

Fig. 20.—Secondary electromagnetic field along line ab at Braxtraesk ore field, with section of drill hole.

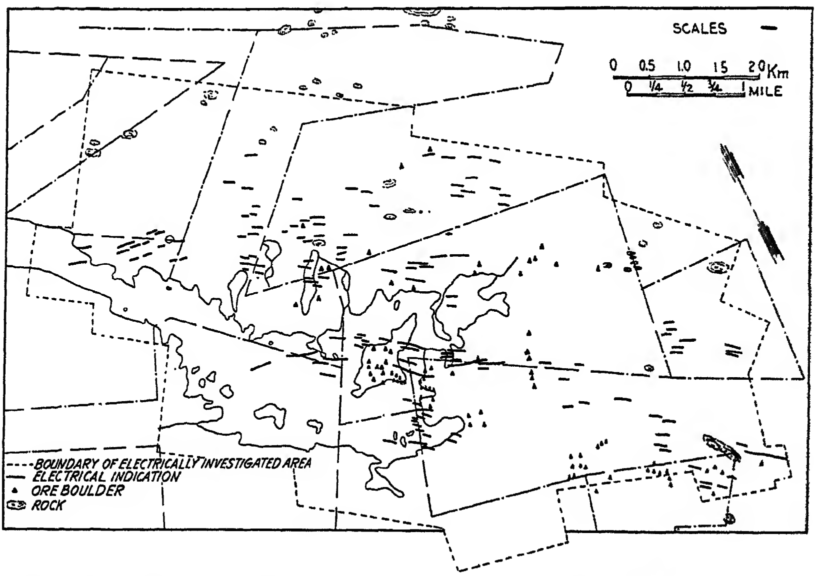

Fig. 21.—Ore boulders and electrical indications at Menstraesk.

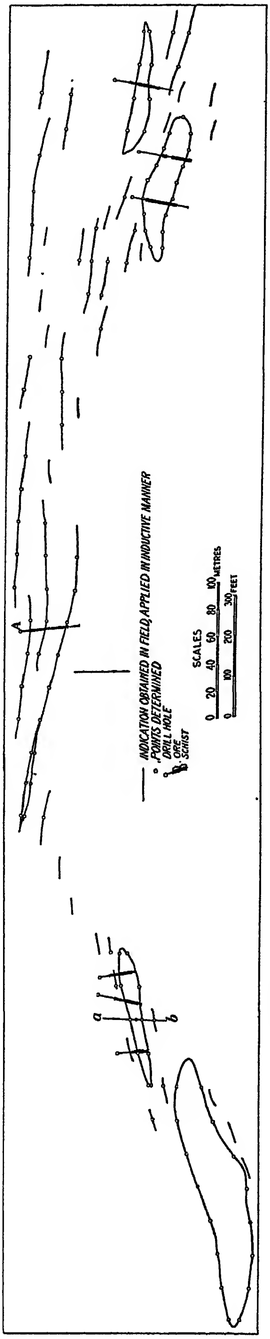

Fig. 22.—Results of drilling and electrical investigations on Lake Menstraesk.

It has been found that most orebodies are situated beneath Lake Menstraesk, an electrical investigation of which was made on the ice during the winter of 1923–24. The lake was frozen over in November, 1923, when the investigations were begun, and in January, 1924, *i. e.*, only about three months after the electrical investigation was begun, the first ore was found by diamond drilling from the ice. The drilling was afterwards continued all winter and gave results, some of which are shown in Fig. 22; Fig. 23 illustrates the secondary field along the line *ab*. The total overburden of earth and water varies between 20 and 60 ft.

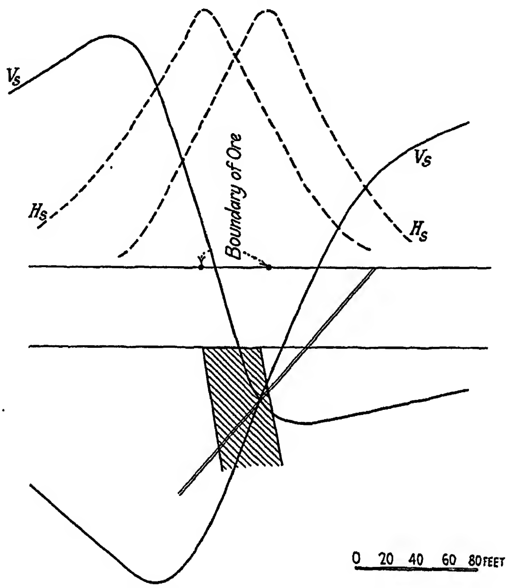

Fig. 23.—Secondary electromagnetic field along line *ab* with section of drill hole.

The orebodies, chalcopyrite and pyrite (3–7 per cent. copper, 40 per cent. sulfur), in Menstraesk lie partly in graphitic schists and partly in the contact between those schists and leptite. This naturally means that it is most difficult to define the position of the orebody. In choosing the indications, the qualitative investigations have been of great use. This can best be seen from the fact that, as shown by Fig. 22, all drill holes, except *A*, struck ore. It should be observed, however, that drill hole *A* was made on indications which, it was predicted, were probably not due to ore. The hole was drilled, however, to obtain a continuous block of claims.

Summary

The Swedish methods of electrical prospecting, the potential and electromagnetic methods, have been independently operated with success

in several countries, resulting in the discovery of a great number of orebodies that were previously unknown. Some of them have proved to be of great commercial value. So far, the best results are the discoveries in Northern Sweden of a large sulfide ore district containing the largest deposits of sulfur, copper, and arsenic hitherto found in Sweden. The methods mentioned are being operated together and, as they are based, partly, on quite different characters of ore minerals, the chance of determining the exact position, as well as the nature of the conductor, will now be considerably greater.

DISCUSSION

E. F. Burchard, Washington, D. C.—How feasible would your method be in a region containing a number of prospects, many of which are being held pending the success of one property and consequent rise in prices?

H. Lundberg.—I think that is a favorable place for these methods if any given deposit of ore minerals proves to have electrical characteristics different from the surrounding orebodies, which can easily be tested.

E. F. Burchard.—Then the deposits may be disconnected?

H. Lundberg.—Yes. We have methods for all kinds of deposits.

E. E. Ellis, New York, N. Y.—Do you think that you must have a reasonably connected orebody, or do you require a series of orebodies one after another with intervening barren areas between them?

H. Lundberg.—It has been possible to find many different types of ore, but sometimes preliminary tests have to be followed by small-scale laboratory tests of just the kind of a deposit that you want to discover. The possibility of finding ore cannot be predicted without making a test of the minerals and looking into the conditions.

E. E. Ellis.—Must you have a particularly large unit mass of ore in order to find it, or can you have a large mass in a series of units?

H. Lundberg.—A rather disseminated body will act similarly to a solid body.

W. Lindgren, Cambridge, Mass.—What would happen in such a place as Ammeberg in Sweden where the ore is almost exclusively zinc, which, I understand, is a very poor conductor.

H. Lundberg.—Zinc blendes sometimes are poor conductors and cannot be traced, but at Ammeberg, especially where we have tried this method, the ore has a conductivity so different from the surrounding

rocks that it can be traced. A preliminary test shows whether tracing is possible.

A. M. BATEMAN, New Haven, Conn.—Through what depth of overburden have these tests been found practicable?

H. LUNDBERG.—A direct answer is difficult. The depth depends on the characteristics of the ore, the size of the ore and the characteristics of the surrounding rocks. In Sweden in places we have been fairly successful down to 300 ft., but there is no reason to believe that would be the lower limit. We do not know the lower limit.

M. C. LAKE, Cleveland, Ohio.—Have you ever done anything with the soft ores?

H. LUNDBERG.—We are working on that now in the laboratory. These types of iron ore you mention are not good conductors. Still, they have a different dielectric constant that makes us think it will be possible to work with them in the future by introducing higher frequency than we are using, but we do not know yet.

B. F. TILLSON, Franklin, N. J.—In reply to some of the questions about the lack of continuity of orebody, I think the idea will be clearer if you realize that in either an electrical field or a magnetic field the presence of bodies of electrical conductivity, or of various magnetic permeability, is made evident by the change in the lines of force in that field. Whether these bodies are connected or not, the variation will be found in those fields. The technologist must be able to interpret the variations in such fields, to determine thereby the conditions of orebodies in the earth and what caused those unusual conditions of electrostatic or electromagnetic fields. It seems as though, with considerable technical experience, this should be possible.

I observed that mention was not made in the paper of the prospects of searching for deeper orebodies by applying this method of research from the low side of diamond drill holes and depths.

H. LUNDBERG.—We are working on the matter of application to depths. It is a very difficult question and requires a long period of laboratory work and many field tests before we can obtain any reliable data.

A. H. ROGERS, New York, N. Y.—We have done considerable electrical prospecting in this country without any results. We feel that this proves there is no ore present in the ground tested. We have also done work in places that on further investigation showed no ore. Some rocks are more highly conductive than other rocks, particularly the basic rocks. In planning the prospecting of an area of mineral ground, if electrical prospecting is done first, the favorable points in that area are immediately

located and, if excavation fails to reveal ore there, then the whole area can be rejected. Thus the amount of work that would be done in prospecting a large area would be materially reduced.

L. C. Graton, Cambridge, Mass.—I take it that Mr. Rogers' remarks are subject to two qualifications: First, that he refers to the kind of ore that will give indication, and I judge there are some kinds of ore that may not; and, second, that the area is barren at least to such depth as the test by this method will extend.

A. H. Rogers.—Dr. Graton is right, but if you are investigating mineral ground, you generally have an idea of the sort of mineral you expect to find. We would not presume to prospect around the Mascot mine in Tennessee for the kind of blende in the Mascot mine, because that is almost as non-conductive as the dolomite in which it occurs. On the other hand, there are certain blendes in the West that have a very high resistance, say 150 to 200 ohms per cu. in., and yet such blendes are so much more conductive even on that basis than ordinary rock, which generally has a resistivity of at least 100,000 or 200,000 ohms, that they can be detected.

H. Lundberg.—Their resistivity is 1,000,000 ohms.

A. H. Rogers.—As to the depth, we do not know the limit. That is a function of the size of the orebody, the conductor, also its conductivity. Disseminated copper ore, for example, would have a very low conductivity and we could not expect to do anything with that. The conducting mineral is distributed in fine particles through the rock entirely unconnected, and the conductivity hardly differs from that of the rock without any mineral in it. Then, if there are laboratory tests of the sort of ore sought, that show any conductivity, you can feel fairly confident at considerable depths.

E. F. Burchard.—This problem resembles that of petroleum geologists. They can tell from structure whether oil *may* be there but not whether it is *actually* there until boring is done. I am wondering whether in mining colleges this work would have to be carried on by highly-trained technicians or could it be introduced in the curriculum and generally understood by an average geologist?

A. H. Rogers.—Success with this method is unquestionably a matter of experience and the technique is constantly developing. My firm has been interested in this method almost 2 years. We have seen considerable advance in the practice but a tremendous amount is still to be done. To the mining engineer much about it is a mystery and some very abstruse electrical principles are involved, but unquestionably interpretation of the results is a matter of practice. Anybody can

understand the resulting maps because they indicate the points where the electrical manifestations are found. It is not a subject that any mining geologist can take up and practice without spending a year or two with Hans Lundberg.

Theodore Zuschlag, New York, N. Y. (written discussion).—Electric prospecting carried on in the United States generally did not come up to expectations placed in it. As the foundation of electrical prospecting is undoubtedly on a sane basis (briefly, it is the practical utilization of the fact that the character of any electric or magnetic field is changed by variations of any electric or magnetic constants within their range), the reasons for these failures must be sought in the fundamental principles of this art. Their study is, therefore, the most essential condition for further development.

Unfortunately, the study of these principles involves several of the most difficult problems of electricity and necessitates the knowledge of higher and highest mathematics, normally not included in the ordinary college training. Mr. Lundberg tries to avoid this difficulty by considering simplified cases and testing their application in the laboratory. Naturally, this method can give only approximate results. Since I became interested in electric prospecting, not as a mining engineer, but as an electrical engineer, it was my conviction that the final solution of this problem would be found only by a complete clearing up of the electrical principles involved. But due to the difficulties mentioned, it was not until the last months that I succeeded in lifting the veil. The results of this work, I regret to state, prove that the theories used as the foundation in the Swedish electric prospecting methods are not free from fundamental objections.

A consideration of the general structure of electric and magnetic ground fields will prove my assertion. An electric generator, grounded at two different points or lines, produces a certain flow of elementary distributed currents through the ground. The currents, flowing through the ground and through the lines connecting the generator with the ground plates, set up magnetic fields which, except in case of direct current, induce stray currents in the ground. For direct current the amount of current flow can be measured at the generator, and its elementary distribution in the ground can be easily calculated. Such a formula in gradient form for line electrodes is given in Mr. Lundberg's paper. These formulae, for their first approximation, depend not upon any conductivity values or conductivity variations; they are purely geometrical functions of the grounding arrangement. Therefore, direct-current electric fields will be disturbed by conductivity variations only in the immediate neighborhood of these disturbances. This fact restricts the possible use of direct current to prospecting for very lightly covered

orebodies that are good conductors. The distribution of alternating current in the ground is covered by other laws.

For grounded arrangements, we have to distinguish between the generator current, the elementary distribution of which can be regarded as corresponding to that of direct current, and an infinite number of stray currents induced by different induction effects. Such are the magnetic field of the generator wire current between the two grounded points or lines, the magnetic field of the stray currents, induced by the magnetic field of the elementary generator ground currents, and so on. This infinite number of current systems superposes and influences each other everywhere, and their resulting strength and distribution cannot be computed by elementary formulae.

Similar conditions prevail for the magnetic fields. For this reason the usual methods of calculating the values of an alternating magnetic field fail here. In order to obtain mathematical expressions for the values of such alternating magnetic or electric fields, one must start from the two famous vector equations of Maxwell, the great English physicist. As this long and difficult computation will later be published elsewhere, I shall give here only the general result of this calculation.

The values of alternating electric or magnetic ground-fields can be expressed by infinite complex series, consisting of a real and an imaginary part, the different factors of which are elliptical and hyperelliptical integrals. These series converge quickly for audio frequencies. The values of the integrals depend on the current strength, the frequency and the layout of the generating arrangement, the surface conditions and the electric constants of the different strata, and other materials of the area within the range of the fields. They vary, of course, with location and direction.

The admissibility and possible application of any alternating-current prospecting method must be judged by these expressions. Their interpretation in regard to equipotential line methods shows at once that these are not generally applicable in alternating fields. The reason is that equal alternating current potential presumes equal intensities as well as equal phase angles in regard to the generating electromotoric force, which conditions are practically never realized at different points in an alternating ground-field. This fact explains the observation that, in measuring equipotential lines in alternating fields, a telephone registers normally only a more or less pronounced minimum within a more or less broad sector. The plotting and interpretation of these minimum lines promise success only when the orebody looked for is very lightly covered and of good conductivity.

The same applies to the electromagnetic determination of the so-called lines of equal magnetic flux. Apart from this fundamental objection, the interpretation of the results of these methods proves very

difficult in any other than a flat country, due to the indeterminable effects of surface conditions. In hilly country indications may be caused by conductivity differences in the ground as well as by slopes, valleys or hills.

The two other methods described by Mr. Lundberg deal with the measuring of magnetic field intensities. One method uses an inductively coupled generating arrangement. In this case no generator ground current exists. Accordingly, the theoretical expressions do not contain this link, but the general structure of the infinite complex series is not changed. Mr. Lundberg bases his theory upon the assumption that the generating arrangement and the orebody form a short-circuited transformer. This theory evidently does not take into account all the necessary conditions, as the influences of the surrounding rock and of the surface conditions are completely ignored. For the other method a formula is given which, however, only slightly approximates the actual conditions and which does not take into account the surface conditions.

Mr. Lundberg's paper states nowhere, how the practical measuring or calculation of the so-called secondary field is carried out. Therefore, I can only assume that it is determined by subtracting the calculated value of the primary field from the measured value of the combination field. This subtraction can be correct, according to the former explanations, only if carried out with the complete complex intensities, because the phase angles of the secondary field vary with the location and even with different directions at one location. As nothing in the paper indicates that this demand was recognized and the intensities determined accordingly, it cannot be expected that the Swedish prospecting methods may be used successfully except for lightly covered orebodies that are good conductors.

Relations of Metalliferous Lode Systems to Igneous Intrusives

By W. H. Emmons,* Minneapolis, Minn.

(New York Meeting, February, 1926)

Introduction

This paper is the second of a series treating the relations of ores of the metals to igneous rocks. In the first paper[1] the general problem was outlined and the normal downward changes in metalliferous lodes were briefly stated, with references to earlier papers treating the relations of ores to depths of their formation. The present paper by a number of examples illustrates the relations treated in the first paper. These examples are chosen chiefly from the larger mining districts because their underground developments are more extensive, but the same relations are shown in maps of scores of undeveloped areas in which the main geological conditions are nearly similar.

Ecology of Metalliferous Deposits

Ore deposits are divided into three groups: 1, Those formed by sedimentary processes; 2, those deposited by meteoric waters; and, 3, those formed in connection with igneous processes. The last group is the most important. It includes the magmatic segregations, which are igneous rocks in the strict sense, and the veins and related deposits, which were formed by precipitation in channels from solutions originating in cooling intrusives. These deposits may be divided into two groups: 1. Those related to intrusives as basic as diorite; and, 2, those related to intrusives more acid than diorite. The second group probably includes more than 95 per cent. of all lode ores. Nearly all of the ores of this group are associated with granitic batholiths or are in positions where it is reasonable to suppose that such a batholith, not yet exposed by erosion, underlies the area containing the deposits.

The batholiths are great masses of deep-seated igneous rocks. When they are intruded most of them are probably as basic as diorite and some

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[1] W. H. Emmons: Primary Downward Changes in Ore Deposits. *Trans.*, **70**, (1924), 964–997.

are more basic. They become more acid by differentiation. The older rocks of the batholith are, for example, peridotites, gabbros, and diorites, and the younger ones are quartz-diorites, monzonites, grano-diorites and granites. This is a matter of observation. The series is established by petrographic studies in many districts. Nearly all lode ores are derived from solutions that leave the batholith when or after differentiation has reached the stage of quartz-diorite or monzonite. The great majority of such solutions are probably expressed from the batholith after it has reached the granodiorite or quartz monzonite stage.

Most batholiths widen downward; that also is a matter of observation. Their roofs are undulating and upon them are many combs and cupolas and these generally bear an important relation to the deposits which, the world over, are grouped around the batholiths or occupy their margins. Fig. 1. (p. 32) is a cross-section of an ideal batholith. Only the main intrusive is shown; dikes, sills, and laccoliths are omitted. Many magmatic segregations, such as iron, nickel, and chromium ores, are parts of laccoliths and a few valuable deposits of silver ore are associated genetically with the larger sills, but these are not treated herein. Many lodes in and near batholiths are associated with dikes. The solutions that deposited them followed the dike magma upward, but the lode is generally later than the dike. The smaller dikes, less than 100 ft. wide, probably do not carry sufficient metalliferous material to supply the ores that form the lodes, and they cool too quickly to yield adequate differentiation products.

Inspection of the batholiths of the earth, which are exposed chiefly as granite and granodiorite masses, and their associated lode deposits, shows a consistent relation. Lode deposits are associated with most of them. A few of the larger ones are essentially barren and some of the smaller ones seem to be barren also, but the total exposures or outcrops of these cover very much smaller areas than outcrops of the larger barren batholiths.

The vast majority of lodes associated with the granitic rocks are features of their roofs. They are near their contacts or make outward from the contacts in the invaded rocks. The interiors are generally barren. This relation, which was shown by Lindgren to exist in California, by Butler and Loughlin to exist in Utah, and by Scofield to exist in the Coast Range of Canada, will hold the world over. Practically all the important deposits in the great granitic masses are at or near their edges, in or near roof pendants, at places where there is reason to suppose that the eroded roof was flat, or in or near rocks that were intruded into the batholith after it had solidified. Of the last, most of the deposits in the deeply eroded batholiths are closely related to a second and much later period of deposition. Certain deposits in the great granitic masses may, however, be related to a late episode of the intrusion, but these examples

of doubtful genesis are very few compared to the great bulk of the lodes that are clearly associated with the roofs.

Preferred Situations for Metalliferous Lodes

The preferred positions near the contacts are in and near the cupolas, the roof combs, or ridges, and in plunging roof-ridges. This refers not only to the lodes themselves, but to the valuable deposits of contact metamorphic origin. This is true only of deposits that lie on or near the high parts of the roofs; tin, tungsten, arsenic, copper, zinc, gold and silver, and some others. Antimony and mercury migrate far from the ridges and cupolas but commonly are in series related to them. Valuable lodes are often found, however, at places where no cupola, comb or tilted ridge is evident. In the deeper parts of batholith's roof, the deposits generally seem to bear little or no relation to the ridges and combs. In the roof pendants they lie in all positions. The writer, after inspection of all the data available to him relating to the many regions showing lodes in roof pendants, discovered no consistent relation whatever to the details of the roof contour.

Although all of the metals, except such as chromium, platinum and nickel, come off of the high places of the batholiths in large amounts, only gold, copper and tungsten are prominent in the lodes filled by solutions expelled from the low points. Moreover in the zonal arrangement, copper deposits are generally nearer the cupolas and high points than the gold deposits are; but when low parts of systems are seen on the slopes of deeply eroded batholiths, the copper deposits are farther from the intrusives than the gold deposits. The majority of such deposits, however, show but little evidence of zoning and in smaller roof pendants no zoning whatever has been recognized.

Classification of Deposits with Respect to Certain Conditions of Their Environment

Metalliferous lodes may be divided into six groups according to their positions with reference to their parent batholiths. Gradations between these, exist, of course, but the six classes afford convenient groups for comparison and all but the first are easily recognized in the field. So many observations are available regarding the shapes of batholiths the world over that one may with some assurance construct an ideal one such as is shown in Fig. 1, a cross-section, and Fig. 2, contours on part of the roof. Only the deposits on the six surfaces are shown in Fig. 1, and Fig. 2 shows only the deposits at or very near the contact. Lode deposits are greatly concentrated at, near and above the cupolas and roof ridges, except those of gold which are found in all positions and are very abundant at low horizons.

The six groups[2] of deposits are:

1. Cryptobatholithic, near hidden batholiths.

2. Acrobatholithic, in and near cupolas or domes.

3. Epibatholithic, on batholiths, near their rims, but below the eroded cupolas or highest points of batholiths. In areas containing these deposits the invaded rocks predominate.

4. Embatholithic, among or between closely spaced batholiths. The invaded rocks predominate and form the frame of the region containing the batholith.

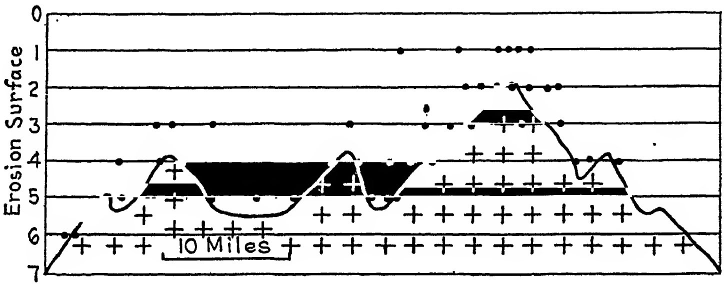

FIG. 1.—CROSS-SECTION OF AN IDEAL BATHOLITH, SHOWING BY DOTS THE MOST FAVORABLE POSITIONS FOR LODE DEPOSITS AND SHOWING SIX STAGES OF EROSION OF THE BATHOLITH.

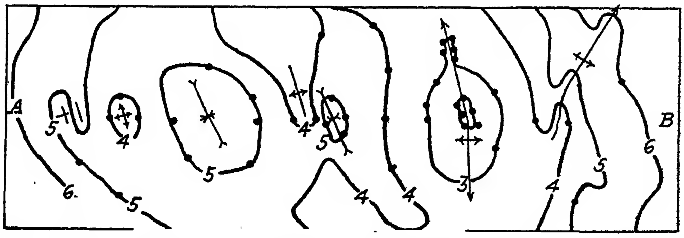

FIG. 2.—PLAN OF ROOF OF BATHOLITH, SHOWING BY DOTS THE MOST FAVORABLE POSITIONS FOR LODE DEPOSITS. THESE LIE ON THE ROOF AND ARE CONCENTRATED AT HIGH PLACES AND IN ROOF PENDANTS.

5. Endobatholithic, in and near roof pendants of large batholiths. The invading rocks predominate and form the frame of the area containing the deposits.

6. Hypobatholithic, in deeply eroded batholiths where even the roof pendants are removed.

Fig. 1 attempts to show how these situations exist at six stages of erosion of a single batholith. Fig. 3 shows the relative amounts of cer-

[2] Greek prefixes in common use are proposed to designate the six groups: 1, Crypto, hidden; 2, acro, summit; 3, epi, on, upon; 4, em, among; 5, endo, within; 6, hypo, down (low, deep).

tain metals developed in each of the six positions. This figure is presented with some hesitation, for there are many metalliferous areas in the earth that are not accurately mapped, but the diagram is believed to be substantially correct and is a convenient way to present the conclusions following the study of a great mass of related data, some of which is reviewed in pages that follow.

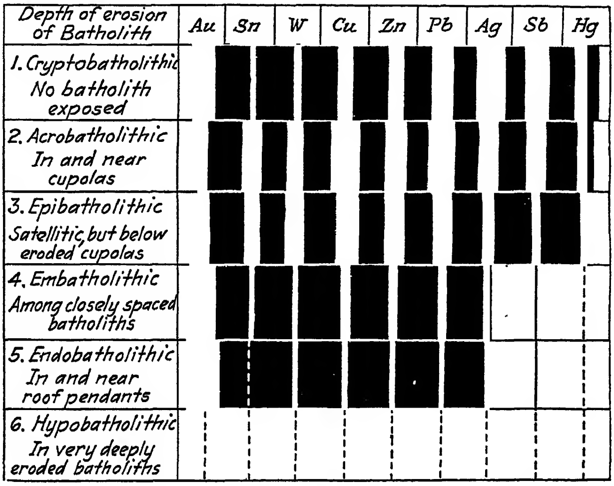

FIG. 3.—DIAGRAM SHOWING THE RELATIVE AMOUNTS OF DIFFERENT METALS DEPOSITED IN LODES AT SIX DIFFERENT POSITIONS ON THEIR PARENT BATHOLITHS.

THE ZONAL ARRANGEMENT OF THE METALS IN LODES

In many districts showing two or more kinds of metalliferous ores, the latter are zonally arranged, often concentrically in successive zones away from an invading rock mass. The horizontal changes from the outer zone toward the center are the same as those noted in single veins from the surface down. In the development of this theory in North America, Spurr has taken the most prominent part. His first paper on the subject appeared in 1907. Important contributions have been made also by Sales, Kemp, Butler, Collins, Lindgren and Loughlin, Billingsley and Grimes, Boutwell, Bastin and Hill, Spencer, Ransome and Calkins, Umpleby, Thompson and McGonigle and others. In Europe, Rastall, Dewey, Hatch, MacAlister, De Launay, Berg and others have treated the subject, and De Launay in his *Traite de Gites Mineraux etc.* (1913) has cited four examples of areas showing the zones. In Australia and Tasmania, Waller, Twelvetrees and Andrews have cited prominent examples. In Asia, Wong has presented diagrams showing the normal series around the intruding granites of southern China. The relation of ore deposits to physical conditions at the time of their formation, which is closely allied to the problem of their zoning, was outlined by Lindgren in a very com-

prehensive way in 1907. A short paper by W. H. Emmons on the genetic classification of minerals followed in 1908.

The metals included in the zones as far as they are named are substantially in similar positions, except the zones of Spurr[3] who names three series. One is connected with basic magmas: chromium, platinum, nickel, copper, (silver) zinc, lead, (silver). Another is connected with intermediate rocks: molybdenum, tungsten, gold, copper, (silver) zinc, lead, (silver.) A third, connected with acid rocks is: molybdenum, tin, tungsten, copper, (silver) zinc, lead, (silver.) Waller, De Launay, Twelvetrees, Rastall, Wong, and most other students name essentially the series: tin, copper, zinc, lead, silver, antimony, although one or more of the metals is omitted in most of the systems.

It is scarcely expected that the zones about all intrusives should show the same order, for the solutions depositing them are not uniform. The metallic sulfides or oxides that are present in great excess should tend to be precipitated first even if they are more soluble than other salts, in accordance with the well known law of concentration effect. Nevertheless inspection of the maps of all areas in the world where the zones have been recorded shows surprising consistency. In groups 2 and 3 (page 4) vertical reversals are almost unknown and consistent reversals of zones as expressed in plan are rare. In group 4, zones are recognized, but they are commonly in reversed order from those shown in 2 and 3. In groups 5 and 6 zones are rare or wanting. Briefly, the zonal arrangement is most clearly expressed in and around cupolas of batholiths; in and around the lower, yet relatively small exposures of batholiths; and in the border regions of large batholiths and embatholithic areas.

A noteworthy example of the last group is the Idaho batholith. The series as given by Thompson and McGonigle,[4] is: 1, gold; 2, silver; 3, lead; 4, copper; the gold deposits occur nearest the batholith. Spurr,[5] discusses these data and restates the series as expressed in earlier papers: 1, gold; 2, copper; 3, lead; 4, silver, and shows how the metals might have been deposited in this series in and around the Idaho batholith if conditions of temperature changed during deposition.

The series, gold in and near the intrusive and copper farther out, is expressed by the zones of Spurr and of Thompson and McGonigle. This series is shown where batholiths are deeply eroded exposing large areas. A well known example is the Sierra Nevada batholith in California where the Foothill Copper Belt parallels the Mother Lode for more than 100 m., lying farther from the intrusive. But around cupolas, gold deposits are commonly found around and outside of the area of copper deposits; the

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[3] J. E. Spurr: The Ore Magmas, 2 (1923), 611.

[4] F. A. Thompson and Fritz McGonigle: Zonal Distribution of Gold, Silver, Lead and Copper Ores in Idaho. *Eng. & Min. Jnl.*, 120 (1925), 216–218.

[5] J. E. Spurr: Ore Zones of Idaho. *Eng. & Min. Jnl.* 120, (1925), 361–362.

situation is reversed. Examples include the deposits of Morenci, Ely, Bingham, and copper and gold deposits northwest of Redding, California. (Figs. 8, 9, 10 and 13.)

For the reader's convenience the series as named in the author's earlier paper will be restated: tin, tungsten, arsenic, bismuth, gold, copper, zinc, lead, silver, gold, antimony mercury. The positions of certain other metals are less well defined. They overlap the zones of metals named but they predominate in two or more zones. Iron oxides of primary (hypogene) origin appear with and below copper in the main. Iron sul-

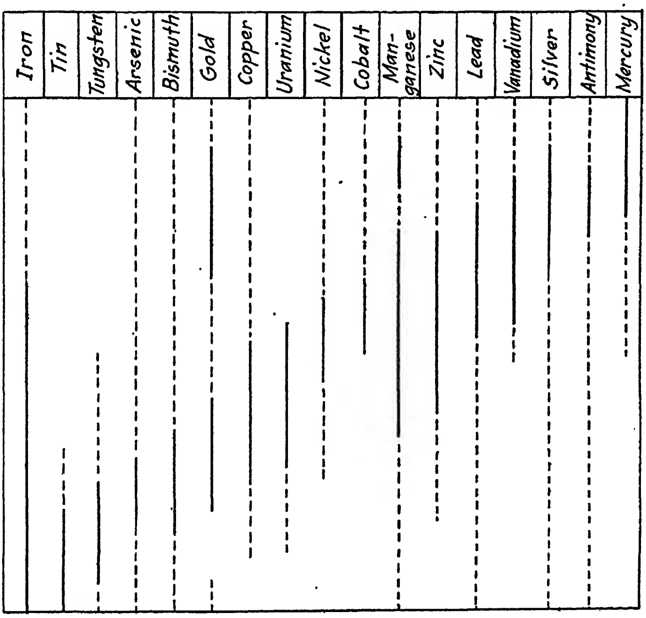

FIG. 4.—DIAGRAM SHOWING THE NORMAL POSITIONS OF METALS IN VEIN SYSTEMS CONTAINING TWO OR MORE METALS.

fide has about the same range and extends upward, generally in smaller amounts, to the highest zone. The other minerals included in Lindgren's persistent group likewise have very wide ranges in the zones. Manganese minerals in lodes appear with copper and extend upward. Rhodochrosite extends nearly to the top of the system and rhodonite generally is more restricted. Nickel is not common in veins; its place is with and above copper, but data are insufficient for a more definite statement. Cobalt is with nickel. Platinum, where present in veins, seems to have the same position as the deeper gold deposits. Uranium is found with both gold and copper, and vanadium in workable amounts is almost restricted in North America to zones of copper, zinc and lead. These relations are tentatively expressed in Fig. 4.

Group 1: Cryptobatholithic Deposits

Deposits of this group are typically developed in the zone of Tertiary deformation that belts the Pacific[6] and extends from the East Indies westward through southern Asia, southern Europe, northwest Africa, to the West Indies and Mexico. This belt, which is 40,000 m. long, is the greatest single ore province of the earth. The districts containing the deposits include Goldfield, Tonopah, Comstock Lode, Appolo, Sado Islands, Taiwan, Waihi, Andes (in part), Lebongstreek, Nagyag and others.

In most of these districts the ore deposits are closely associated with andesites, dacites and shallow-seated porphyries. Only rarely are typical zones developed, although there are few, if any, important reversals of the normal series passing downward in individual veins. In many districts the metallization is concentrated in areas 3 or 4 m. square. The inference that these deposits lie above high points of unexposed batholiths is pure speculation, yet it is supported by many facts, particularly by their geological and geographic surroundings,[7] and by the group of hypotheses developed from the study of mining districts where erosion has gone only slightly deeper than in areas containing deposits of group 1.

In a few districts dikes are closely spaced in certain centers suggesting that a high point of a deep seated intrusive lies below, and rude zonal arrangements in the normal order are shown around the areas of closely spaced dikes. Examples include the Northern Black Hills, Pine Creek district and Battle Mountain district in Nevada.

Northern Black Hills

The deposits of the northern Black Hills are believed to form a cryptobatholithic system. The metallized area (Fig. 5). is made up of pre-Cambrian schists surrounded by sedimentary rocks that dip away from the schists. Porphyries, probably early Tertiary, intrude both the pre-Cambrian and later rocks. The larger number of the deposits probably were formed at about the time of the intrusion of the porphyries and it is believed that the metalliferous solutions were derived from a deeper seated igneous mass that supplied the porphyries.[8] The most valuable deposits are those of the Homestake mine which replace folded calcareous

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[6] In 1905 Spurr called attention to certain similar characteristics of deposits that are associated with the Pacific volcanic belt: Geology of the Tonopah Mining District. U. S. Geol. Survey *Prof. Paper* 42, (1905), 275–287.

[7] W. H. Emmons: The Major Hinge Zone of Tertiary Deformation and Some of Its Precious Metal Deposits. Int. Geol. Cong., Brussels 1926. (In press.)

[8] J. D. Irving, S. F. Emmons and T. A. Jagger: Economic Resources of the Northern Black Hills. U. S. Geol. Survey *Prof. Paper* 26 (1904), 1–222.

S. Paige: Central Black Hills Folio (No. 219), U. S. Geol. Survey (1925).

J. O. Hosted and L. B. Wright: Geology of the Homestake Orebodies and the Lead Area of South Dakota—I. *Eng. & Min. Jnl.*, **115** (1923), 793–799..

beds in the pre-Cambrian schists. Irving and Paige concluded that the deposits were formed in pre-Cambrian time and this theory has long been accepted, but Hosted and Wright have stated their belief that the deposits are Tertiary, for which they find support in the close association of the ores with the rhyolitic porphyry dikes of Tertiary age.

The Tertiary group of deposits presents a series zonally arranged. These are: 1, Deposits of siliceous ores with gold, tungsten and arsenopyrite, including the gold and tungsten ores in the Cambrian at Lead, similar deposits of the Yellow Creek area 2 m. south of Lead, and the gold and tungsten deposits of Two Bit Creek. 2, Siliceous gold ores with some silver, in general replacing limestone, including the deposits of Garden, Squaw Creek, Ruby Basin, Portland and Strawberry Gulch. With the same group are placed the gold deposits and sphaleritic gold

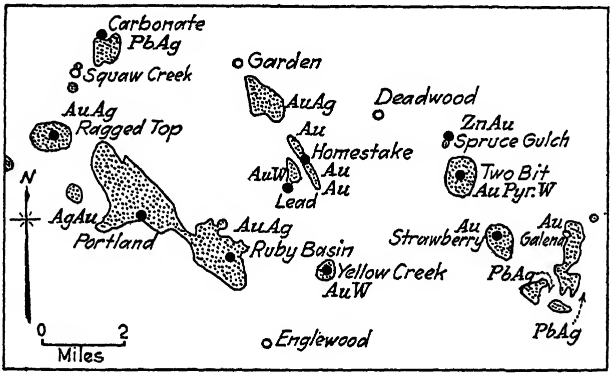

FIG. 5.—MAP SHOWING POSITIONS OF METALLIZED AREAS OF NORTHERN BLACK HILLS (STIPPLED). THE ZONES ARE: 1, GOLD AND TUNGSTEN—LEAD, YELLOW CREEK, TWO BIT; 2, GOLD AND SILVER—GARDEN, RAGGED TOP, PORTLAND, RUBY BASIN, STRAWBERRY; 3, SILVER AND LEAD—CARBONATE AND GALENA. (DATA FROM IRVING, PAIGE, WRIGHT, HOSTED AND OTHERS.)

deposits of Spruce Gulch, 1½ m. southeast of Deadwood. 3, Still farther from the central gold-tungsten area and outside the gold-silver belt, are the silver-lead deposits of Carbonate on the northwest and those of the Galena district on the southeast edge of the mineralized area.

The Homestake deposits are not included in the zones mentioned. They carry gold and arsenopyrite, but are practically without tungsten. They are of the high-temperature type, however, as the gangue carries heavy silicates.

Battle Mountain, Nevada

The Battle Mountain district, Nev., a few miles southwest of Battle Mountain station on the Union Pacific Railway, is an area of Paleozoic rocks intruded by dikes and sills of granite porphyry.[9] The district

[9] J. M. Hill: U. S. Geol. Survey *Bull.* 594 (1915) 64–91.

(Fig. 6) probably lies above a batholith, and Copper Basin is probably above a high point in the roof. It is an area of closely spaced dikes marked by the deposition of copper ores. About 2 m. west of Copper Basin lead-silver ores are found and farther west are ores of antimony. About 3 m. to the north gold ores appear. In another center southwest of Copper Basin copper ores are found at the Virgin mine; farther north near Galena, gold-silver and lead deposits appear. Antimony ores are found still farther north at Antimony King.

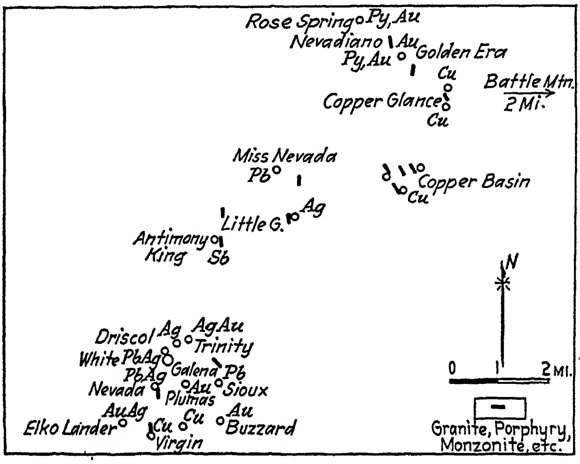

FIG. 6.—SKETCH SHOWING THE METALLIZED AREA NEAR BATTLE MOUNTAIN, NEV. THE DISTRICT IS PROBABLY CRYPTOBATHOLITHIC AND SHOWS THE SERIES: 1, COPPER; 2, LEAD SILVER; 3, ANTIMONY. (DATA FROM HILL.)

Yellow Pine, Nevada

In the Yellow Pine[10] district in southern Nevada, Paleozoic and Mesozoic sedimentary rocks are capped by lavas and intruded by quartz monzonite and granite porphyry dikes and sills. No large intrusives and almost no contact metamorphism are near the porphyries. The deposits (Fig. 7) are believed to be related to a hidden batholith with the high point of the batholith probably below a line connecting the Boss, Keystone and Red Cloud mines since in this area the dikes are closely spaced. The deposits are gold and gold-platinum ores in or very near the porphyry; some copper, zinc and silver-lead ores in the central intruded areas; and lead, zinc, vanadium and some copper ores farther from the center. The deposits are not clearly separated into zones and there is evidence of some overlapping of copper deposits.

[10] J. M. Hill: The Yellow Pine Mining District, Clark County, Nevada. U. S. Geol. Survey *Bull.* 540*f* (1914) 223–274. A. Knopf: A Gold-Platinum-Palladium Lode in Southern Nevada. U. S. Geol. Survey *Bull.* 620*a* (1915), 1–18.

GROUP 2: ACROBATHOLITHIC DEPOSITS

As already stated, lode deposits in the main are features of the roofs of batholiths. The cupola is a high point in the roof. It is in general larger than the dike and represents a place where the magma rose to the surface through stoping or by some other means displaced the roof. The rocks are either granular or the deeper seated porphyries. The regions

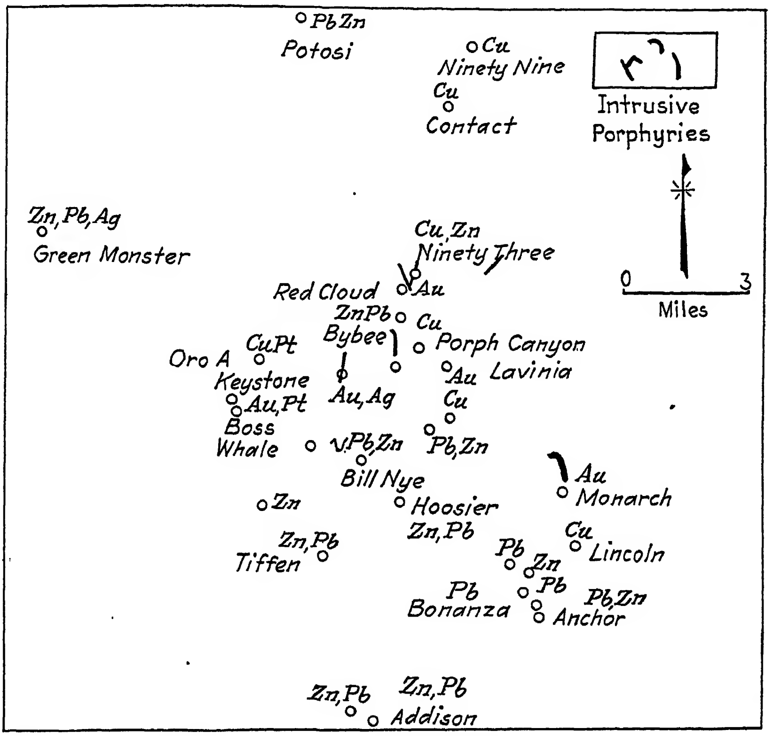

FIG. 7.—SKETCH SHOWING THE METALLIZED AREA OF YELLOW PINE DISTRICT, NEVADA. THE DEPOSITS CLUSTER AROUND CLOSELY SPACED QUARTZ MONZONITE AND PORPHYRY DIKES. THE SERIES IS: 1, GOLD, GOLD-PLATINUM; 2, COPPER, ZINC, LEAD AND SILVER; 3, LEAD, ZINC, VANADIUM. THERE IS SOME OVERLAPPING. (DATA FROM HILL AND KNOPF.)

in, near and above the cupolas are probably the most favorable of all positions for deposition of ores other than gold. Gold is unique in that it occurs low on the walls of batholiths in very much larger amounts than other metals.

The cupola may be regarded as the top of a dome on the roof of a batholith. The metal-bearing fluids which arise from the cooling igneous mass tend to move to the high points much as oil and gas rise in domes and anticlines. The methods of segregation are comparable, as both are brought about by upward movements of liquids and gases, but the con-

ditions are different, in that oil and gas rise through more nearly uniform openings, whereas the metal-bearing solutions pass through highly localized channels that are generally due to fracturing. There is also this essential difference: The hydrocarbons tend to collect in largest amounts in the deeper basinward folds, whereas the metals seem to rise in very large amounts to the highest cupolas, for the deposits in many of the foremost mining districts cluster around an isolated stock or group of small stocks of intrusive rocks.

Concentric and Eccentric Series of Zones and Their Linear Extensions

Around cupolas the metals are commonly arranged concentrically in zones. One group of metals is often found nearest the outcrop of the cupola and another or other groups are farther away. The rock outcropping in the dome, however, probably did not supply the metals for the deposits, as it is commonly fractured and metallized itself. The source of the ore is deeper—an unexposed mass below the cupola—yet is probably part of the same rock unit or a part that was intruded at the same time. Concentric series around small igneous bodies, which are believed to represent tops of domes of larger underlying masses, include those of Ely, Nev.; Morenci, Ariz.; Bingham, Utah; Bisbee, Ariz.; White Pine, Nev.; Redding, Calif.; Cligga Head and Saint Agnes, Cornwall; Tavistock, Cornwall and Devon; the Erzgebirge, Saxony, and many others. This arrangement is very common. Where two or more metals are present the zones are generally exhibited around cupolas. This arrangement is also shown where erosion has gone deeper and larger areas of the batholiths are exposed, but it is rarely shown in endobatholithic areas in and near roof pendants.

By inspection of many areas including concentric zones a series may be established which is regarded as the normal series. The locus of the deposits of the metals that normally are most deeply seated may be regarded as the epicentrum or the place assumed to be directly above a high point of the roof of the batholith. In a few districts the epicentrum is outside of the cupola or the most prominent outcrop of the rock to which the areas are genetically related. At Tintic, Utah, the epicentrum is a linear area extending northwest from the Carisa stock. The metallization is eccentric with respect to the Silver City stock. The zones are in the normal order but are off center to the north of the outcrop of the main intrusive stock. At Bingham, Utah, the deposits are concentric with respect to the Bingham stock, but eccentric with respect to the Last Chance intrusive. It is believed that the epicentrum lies in the area of the Utah Copper disseminated deposits.

In certain districts a group of deposits is found around two cupolas and in the areas between them. The centrum which is assumed to be a high area of the batholith is probably a ridge and the epicentrum is a long

and relatively narrow area above the ridge including the two cupolas. Thus, in Cornwall, tin deposits are grouped around the small cupolas of St. Agnes and Cligga Head (Fig. 16, p. 56). Tin veins are found also in the area between these cupolas and the tin-bearing area extends far from the cupolas at either end of a line joining them and extended beyond to the northeast and southwest. Outside of the tin-bearing area are ores with zinc, silver and lead. This arrangement seems to hold also in the Tavistock district, Devon and Cornwall, (Fig. 17) where deposits with tin, tungsten and arsenopyrite are found in a belt striking east and including the Gunnislake and Kit Hill Stocks. On either side of this central zone with tin, arsenic and copper, are valuable deposits of silver and lead.

Clifton-Morenci District, Arizona

The Clifton-Morenci[11] district is in southeastern Arizona (Fig. 8), about 20 m. from the New Mexico line. The rocks include pre-Cambrian

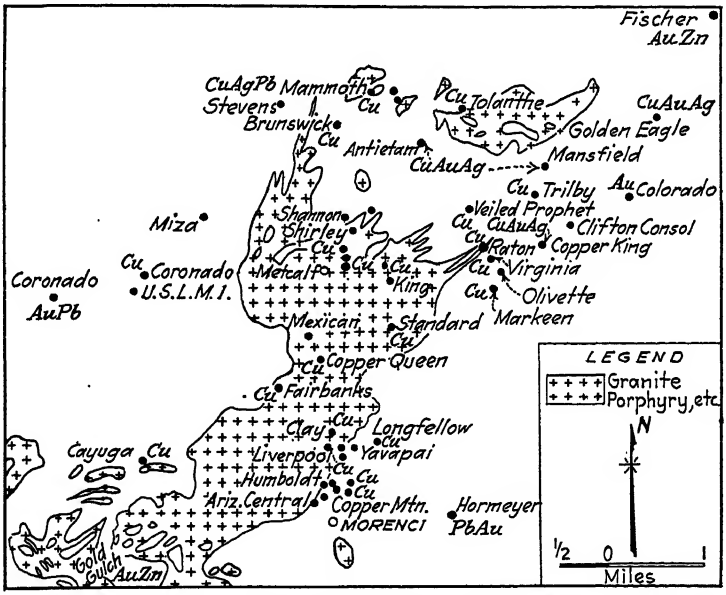

FIG. 8.—SKETCH SHOWING THE METALLIZED AREA OF MORENCI DISTRICT, ARIZ. COPPER ORES WITH SOME ZINC ARE FOUND IN AND NEAR THE INTRUDING ROCK, AND FARTHER AWAY ARE DEPOSITS OF ZINC, LEAD, GOLD AND SILVER. (DATA FROM LINDGREN.)

schists and granites, Paleozoic and Mesozoic sedimentary rocks and granitic porphyries which intrude the Mesozoic and older rocks. The district is surrounded by Tertiary lavas. The granitic intrusives include granite and other porphyries which outcrop in a belt 11 m. long and 1 or

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[11] W. Lindgren: The Copper Deposits of the Clifton-Morenci District, Ariz. U. S. Geol. Survey *Prof. Paper* 43, (1905), 1–364.

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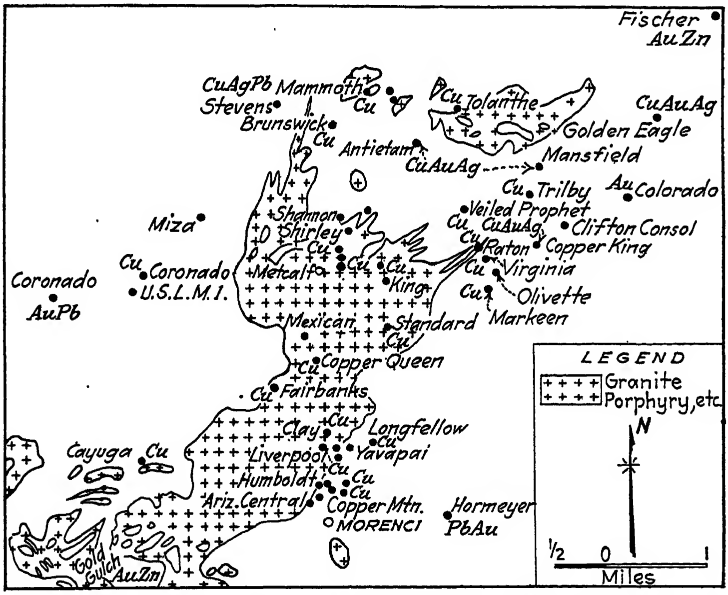

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[11] W. Lindgren: The Copper Deposits of the Clifton-Morenci District, Ariz. U. S. Geol. Survey *Prof. Paper* 43, (1905), 1–364.

2 m. wide that strikes northeast, and probably represent the outcrops of a deep seated batholith which has been unroofed by erosion. The deposits include disseminated enriched copper ores in the porphyry, contact metamorphic and other replacement deposits in the invaded rocks near the porphyry, and veins in porphyry and in invaded rocks. The copper deposits are in or near the porphyry. Zinc is present in some of the copper deposits. Farther from the porphyry, deposits with zinc, lead and precious metals are found. In certain veins and vein systems the metallization changes on strike.

The Metcalf district is at the north end of the largest intrusive mass. The deposits are copper ores and include some of the chief orebodies in the Clifton-Morenci district. The orebodies are disseminated enriched chalcocite ores in porphyry and irregular masses in shale and limestone partly altered to garnet rock. The principal primary ore minerals are chalcopyrite, pyrite and sphalerite.

To the east of Metcalf, copper ores are found in the main intrusive. About half a mile to the northeast of the main porphyry mass a vein system strikes northeast away from the main intrusive. At the mines of Markeen Mountain, copper is the only valuable metal, but to the northeast about half a mile in the Copper King, the ores carry also gold. The Virginia and Trilby veins north of the Markeen lodes also carry copper. In the Colorado vein 1¼ m. northeast of the Copper King, a gold ore vein follows a porphyry dike. The Fischer mine, 1¼ m. northeast of the Colorado, worked zinc ore.

The Coronado vein is about 2 m. west of Metcalf. The vein follows along a fault and is about 1 m. long. On its eastern end only copper ore is found. The most westerly croppings of the vein carry lead and some gold.

At the town of Morenci, the main porphyry intrusive is in contact with Paleozoic limestone and shales. The chief deposits of the Clifton-Morenci district are found in this area. These deposits are disseminated enriched copper ores in the porphyry near the contact with sedimentary rocks, and contact metamorphic and other replacement deposits in the sediments near the porphyry. A complicated system of dikes makes out from the main porphyry mass and some of these also contain disseminated deposits of copper ore.

The Humboldt mine of the Arizona Copper Co. worked large disseminated deposits in the porphyry at the edge of the main mass. Deposits of disseminated ore in the porphyry mass or in the dikes are found also in the Yavapai, Ryerson, Arizona Central and other mines, and large deposits in limestone and shale are found in the Longfellow, Detroit, Manganese Blue, Joy, Montezuma, and others. In the unaltered ore the principal minerals are chalcopyrite, pyrite and sphalerite. In the deposits in limestone and shale, magnetite and hematite are present.

Chalcocite is an abundant alteration product, particularly in the ores in porphyry. These ores carry practically no gold or silver.

In the Gold Gulch district, about 2 m. southwest of Morenci, near the edge of the porphyry mass, small gold veins are found in porphyry and in sedimentary rocks near the porphyry. At the Hormeyer mine 1¼ m. east of Morenci, a vein following a porphyry dike in sedimentary rocks has been worked for lead ore that carries a little gold and copper. The series in the Morenci district is: 1, copper; 2, lead, silver and gold.

Ely, Nevada

In the Ely[12] district in eastern Nevada (Fig. 9), Paleozoic sedimentary rocks are intruded by monzonite and monzonite porphyry in a belt 8 m. long that extends west from the town of Ely. As stated by Spencer, copper ores disseminated in porphyry are found at the Ruth, Liberty, Tonopah, Morris-Bunker Hill, and Veteran mines. These deposits, which are shown as stippled areas in Fig. 9, occupy the epicentral zone of the district. They carry much iron sulfide and little gold and silver.

Almost surrounding the area of copper ores is a belt of silver-lead and silver-gold ores, which before the great copper deposits were discovered were the chief ores worked in the Ely district. In this outer belt the

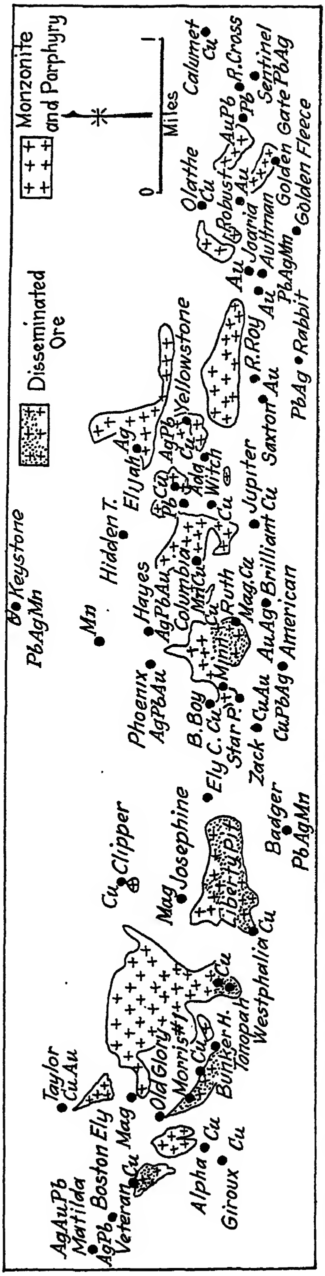

FIG. 9.—SKETCH SHOWING THE METALLIZED AREA OF THE ELY DISTRICT, NEVADA. PORPHYRY STOCKS ARE INTRUDED IN PALEOZOIC LIMESTONE AND SHALE. AREAS OF DISSEMINATED COPPER ORE ARE STIPPLED. THE SERIES IS: 1, COPPER; 2, LEAD, SILVER AND GOLD, AND MANGANESE. (DATA FROM SPENCER, PARDEE, AND JONES AND FROM COMPANY REPORTS.)

[12] A. C. Spencer: The Geology and Ore Deposits of Ely, Nev. U. S. Geol. Survey *Prof. Paper* 96 (1917), 1–181.

A. C. Lawson: Univ. of Calif., Pub. Dept. Geol., 4 (1906), 287–337.

precious metals were more important than lead. The mines of the outer group include the Matilda and Boston Ely (Emma) at its western margin, the Keystone on the north side, and the Badger, American, Brilliant, Saxton, Rabbit, Aultman, Golden Gate, Golden Fleece, and Sentinal on the south side of the copper belt. Manganiferous[13] ores are found also in the area surrounding the copper deposits. The zones from the central axis of metallization outward are: 1, copper ores with iron and a little gold and silver; 2, silver, lead, gold and manganese.[14]

Shasta County, California

In Shasta County,[15] California, (Fig. 10) a copper-gold belt extends in a rude crescent 35 m. long from near Redding to Kennett, then northeast to Bully Hill and east to the Afterthought mine. In this area Mesozoic and older sedimentary rocks and lavas were intruded by granitic rocks near the beginning of the Cretaceous. The granitic rocks include alaskite and quartz diorite which form a complicated igneous series.

Along a belt of quartz diorite about 16 m. northeast of Redding, contact metamorphic deposits of copper ore and magnetite deposits are found. The main deposits, however, are veins and great replacement deposits of chalcopyrite ore in fractured zones in the various rocks to the west and east of the quartz diorite mass. A great area of fine-grained intrusive rocks formerly regarded as (Balaklala) rhyolite is made up of many different intrusives which Graton groups as alaskites and alaskite porphyry. Hershey recognizes seven separate intrusives in the alaskitic complex. This mass which lies northwest of Redding contains the largest copper mines and many gold lodes. Detailed maps are available only for the portion near the Iron Mountain (Mountain Copper) mine where the copper deposits of Iron Mountain and Hornet mines are in or near the third series of alaskite porphyries. In alaskites and in other rocks around this mass, to the east, south and west, gold lodes are found. To the north are the copper deposits of the King Copper, Spread Eagle, Balaklala, Shasta King, Mammoth, Golinski, and Summit mines. To

[13] J. T. Pardee and E. S. Jones, Jr.: U. S. Geol. Survey *Bull.* 710 (1920) 208–242 (1920).

[14] On Fig. 9 certain sills are not shown. Probably no ores are directly connected in genesis with these sills.

[15] J. S. Diller: Redding Folio (No. 138), U. S. Geol. Survey (1906).

L. C. Graton: The Occurrence of Copper in Shasta County, Calif. U. S. Geol. Survey *Bull.* 430*b* (1910), 71–111.

H. G. Ferguson: Gold Lodes of the Weaverville Quadrangle, Calif. U. S. Geol. Survey *Bull.* 540*a* (1914), 22–79.

O. H. Hershey: The Geology of Iron Mountain. *Min. & Sci. Press,* **111** (1915), 633–638.

A. C. Boyle: The Geology and Ore Deposits of the Bully Hill Mining District, California. *Trans.*, **48** (1914), 67–115.

the west of this belt of copper deposits, gold lodes are found and in the French Gulch region these are closely spaced. The copper and gold

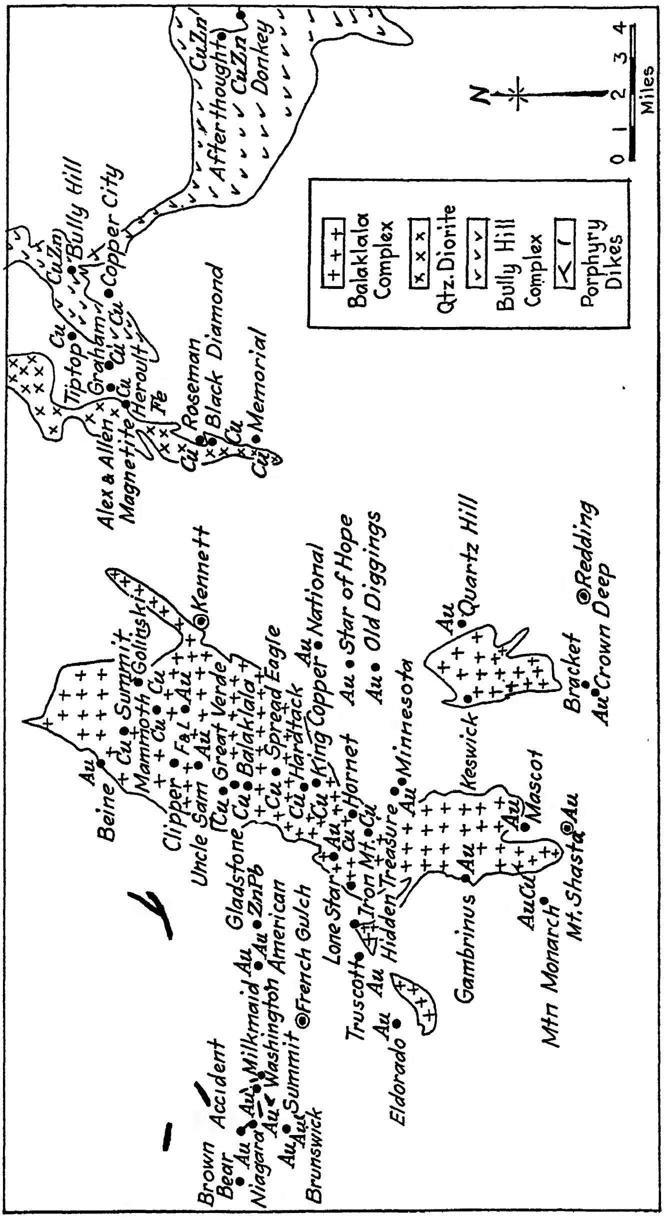

FIG. 10.—Sketch showing relations of Shasta County copper and gold deposits to igneous rocks. The series in the area northwest of Redding is copper, gold. (Data from Diller, Graton, Hershey, Ferguson, Boyle, and others.)

deposits are approximately concentric with respect to part of the alaskitic complex, but that part cannot be accurately designated. Copper deposits

lie in the main alaskitic complex, gold deposits occur near its border near the south end and near either side of the complex. The series is a normal one for the cupola type of metallization: 1, copper; 2, gold.

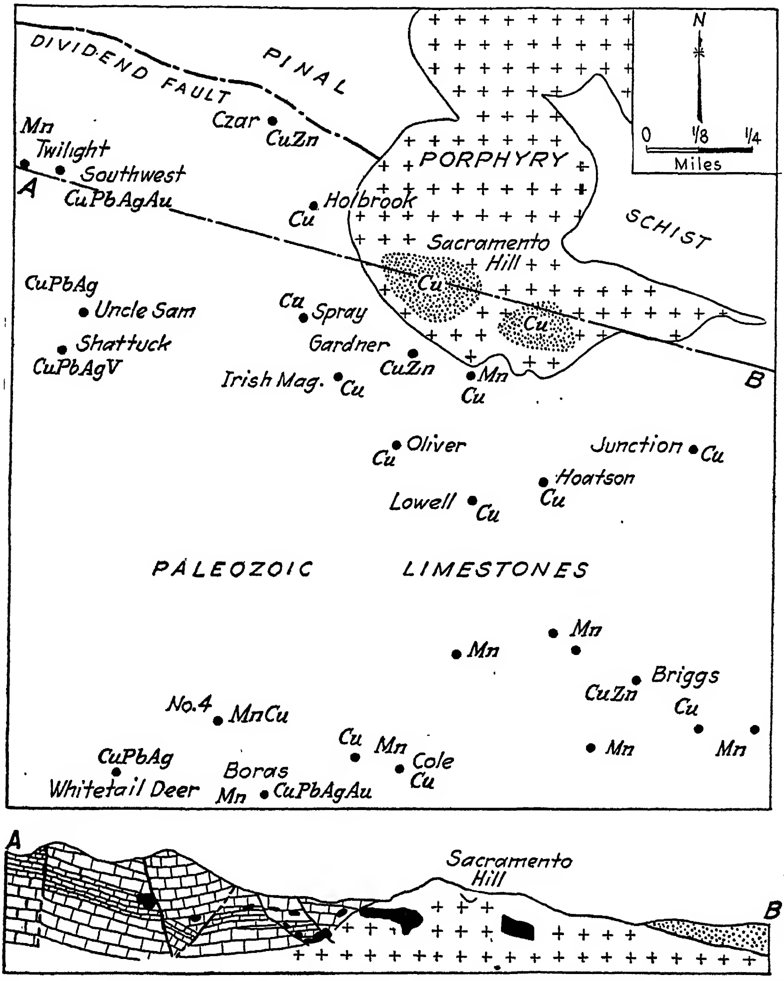

Fig. 11.—Plan and section of Bisbee, Ariz., showing limestone intruded by granite porphyry. Disseminated copper deposits are found in the porphyry. The main copper zone in limestone laps around the porphyry to the west and south. Farther from the porphyry are many valuable copper deposits and some of lead, silver and manganese. Vanadium ores also are found in this zone. (Data from Ransome, Tenney, Bonillas, Feuchère, Prouty, Allen, Butler, and others.)

In the Bully Hill, or Copper City, district a complex of igneous rocks, which Diller mapped as the Bully Hill rhyolite, is made up of various rocks, one of the latest intrusives, according to Boyle, being a series of wide alaskite intrusive bodies. The chief copper ores are great replace-

ment deposits near these intrusives. The Afterthought and Donkey mines, 9 m. southeast of Bully Hill are also in the Bully Hill complex. In this area some of the chalcopyrite ore carries considerable sphalerite and in certain deposits zinc is the principal metal.

Bisbee, Arizona

At Bisbee a mass of granite porphyry breaks through a fault between Pinal schist and Paleozoic limestone. It forms the summit of Sacramento Hill (Fig. 11) east of Bisbee and its eroded surface is covered to the southeast by Cretaceous beds. At the south border of the porphyry great bodies of disseminated chalcocite ores in porphyry have been developed in a zone about a half mile long and half as wide. West and south of the porphyry mass is a series of Paleozoic limestones that dips at low angles to the southeast and is extensively faulted and intruded by dikes radiating from the porphyry mass. Adjoining the porphyry a great series of copper deposits replacing the limestone extends from the Czar mine to the Junction about 2 m., and forms a curved mass following the porphyry contact of Sacramento Hill. Mineralization[16] is so nearly continuous that a projection on a plan shows a nearly solid pattern. This ore contains practically no valuable metals other than copper, although a little zinc is found at the Czar and Irish Mag mines. The orebodies in general follow the beds and are at three chief horizons in the limestones. The ore is deeply oxidized and enriched by chalcocite but on lower levels chalcopyrite becomes prominent. Outside of the copper zone, in general about 4000 ft. from the stock, is a zone in which copper predominates but in which lead-silver and manganese ores also have been worked. The mines of the group include the Southwestern, Uncle Sam, Shattuck, Whitetail Deer, Nighthawk, Boras, and others. Vanadium[17] occurs in the Shattuck mine.

White Pine, Nevada

White Pine[18] (Hamilton) district (Fig. 12), 30 m. west of Ely, is an area of Paleozoic sedimentary rocks intruded by granodiorite. In limestone near the granodiorite, are veins and irregular deposits of chalcopyrite and bornite ore with garnet gangue. A few deposits are found also in the granodiorite. A mile or two east of the copper belt a parallel belt of lead-silver ores is found. The deposits are veins and irregular replace-

[16] F. L. Ransome: The Geology and Ore Deposits of the Bisbee Quadrangle, Arizona. U. S. Geol. Survey *Prof. Paper* 21 (1904), 1–168.

Y. S. Bonillas, J. B. Tenney and L. Feuchère: Geology of the Warren Mining District. *Trans.*, **55** (1916), 284–355.

[17] M. A. Allen and G. M. Butler: Univ. of Ariz. *Bull.* **115** (1921), 13.

[18] A. Hague: Descriptive Geology. Geological Exploration of the 40th Parallel, **3** (1877) 409–443.

W. S. Larch: Mining at Hamilton, Nevada. *Mines and Minerals*, **29** (1909), 521–523.

ments. The ore from this belt, according to Larsh, yielded nearly 100,000 tons lead and $6,000,000 silver. East of the lead-silver belt is one with silver deposits which occupy veins and bedding planes in limestone raised to form a dome. Before erosion and when the ores were formed the limestone containing the ore was capped by shale. Chlorides were abundant and the belt has produced probably about $16,000,000 silver.

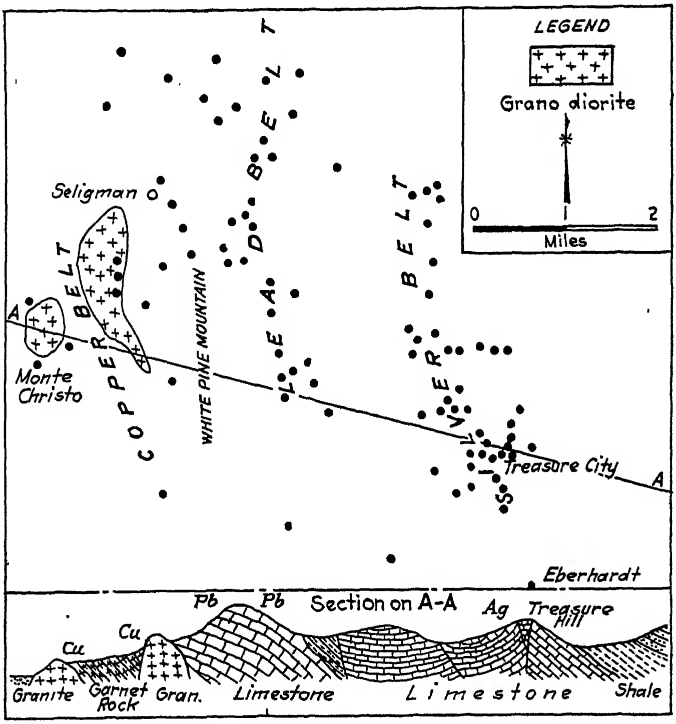

FIG. 12.—SKETCH OF WHITE PINE (HAMILTON) DISTRICT, NEVADA, SHOWING A BELT OF COPPER ORES IN THE GARNETIZED AREA NEAR GRANODIORITE, A BELT OF LEAD ORES WITH SOME SILVER ABOUT 2 M. EAST OF THE COPPER BELT, AND A BELT OF SILVER ORES ABOUT 4 M. EAST OF THE COPPER BELT. (DATA FROM HAGUE, LARSH AND OTHERS.)

Bingham, Utah

The Bingham[19] district, Utah, (Fig. 13), which lies 20 m. southwest of Salt Lake City, is, according to Boutwell and Keith, an area of Carbonif-

[19] J. M. Boutwell and A. Keith: The Economic Geology of the Bingham Mining District, Utah. U. S. Geol. Survey, *Prof. Paper* 38 (1905), 1–413.

B. S. Butler and others: The Ore Deposits of Utah. U. S. Geol. Survey *Prof. Paper* 111 (1920), 340–362.

R. N. Hunt: The Ores in the Limestones at Bingham, Utah. *Trans.*, (1924) **70**, 856–883.

J. J. Beeson: The Disseminated Copper Ores of Bingham Canyon, Utah. *Trans.*, (1916) **54**, 356–401.

erous quartzite with included limestone beds that in general dip northwest at high angles. These beds are faulted and intruded by monzonite and monzonitic porphyry. The main intrusives are the Bingham stock at Upper Bingham, and the Last Chance stock, southwest of it. These

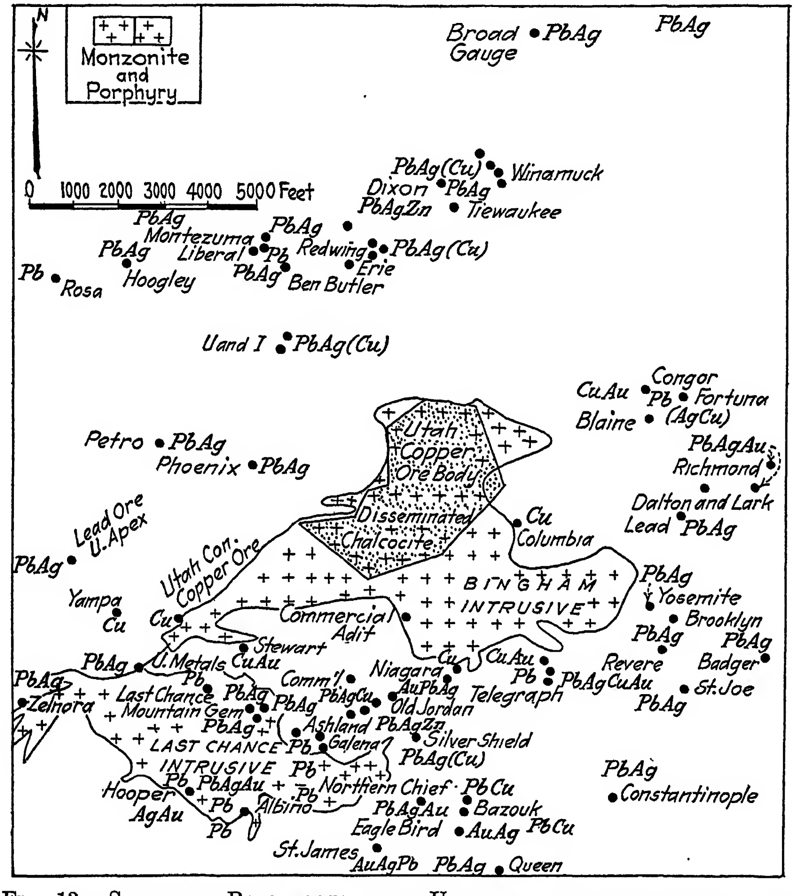

FIG. 13.—SKETCH OF BINGHAM DISTRICT, UTAH, SHOWING THE DEPOSITS CONCENTRIC WITH RESPECT TO THE BINGHAM INTRUSIVE AND ECCENTRIC WITH RESPECT TO THE LAST CHANCE INTRUSIVE. THE SERIES IS: 1, COPPER ORE IN THE BINGHAM STOCK (UTAH COPPER); 2, COPPER ORE AND LEAD ORE NEAR ITS CONTACT—OHIO, COMMERCIAL, TELEGRAPH, NIAGARA, OLD JORDAN, BROOKLYN, ETC.; 3, LEAD-SILVER ORE FARTHER AWAY—FORTUNA, RICHMOND, GALENA, QUEEN, LAST CHANCE, UTAH METALS, UTAH APEX, TIEWAUKEE, ETC. (DATA FROM BOUTWELL, HUNT AND BUTLER, AND FROM COMPANY REPORTS.)

intrusives send off many sills and dikes and the Bingham mass was once regarded as a laccolith, but drilling to a depth of 1500 ft. has shown no floor and it appears probable that the Bingham intrusive is a stock and the top of a batholith.

The Bingham district contains some of the greatest copper and lead-silver deposits in the United States. The deposits are disseminated copper ores in porphyry, copper ores, and lead-silver ores in limestone and quartzite. The ores belong to a single series and are concentric with respect to the Bingham stock, which probably is the summit of the deep seated intrusive that doubtless supplied the solutions that deposited the ores. The ores are eccentric with respect to the Last Chance stock, which is a mile or more off center to the southwest.

The disseminated sulfide copper ores in porphyry are shown on Fig. 13. They are confined to the Bingham stock and extend to an average depth of 530 ft. These ores are found in and near innumerable small fractures in the porphyry and owe their value largely to chalcocite enrichment. Outside of the Bingham stock, the chief deposits are sulfide copper and lead ores with precious metals that replace limestone lenses in the quartzite.

On the east edge of the stock, copper ores are mined in fractured quartzite in the Columbia mine and in other mines of the Ohio Copper Co. Sulfide ores replacing limestone dipping northwest are found in a series of deposits in the Telegraph and Niagara mines south of the intrusive. The ore carries silver, lead, gold and copper, the latter becoming prominent in the lower limestone. In the Old Jordan, ores or lead, silver, copper and gold are found in the limestone. Farther west in this belt at the Ashland and Galena mines, lead-silver ores are mined in quartzite. This zone of lead ores extends into and across the Last Chance intrusive. Thus there is a belt of deposits extending from the Old Telegraph mine to the Albino, mainly in the Jordan limestone, but locally in quartzite and porphyry. Lead and copper ores are found in this belt in large amounts, the copper decreasing away from the Bingham stock and the proportion of lead increasing.

South of this belt veins of lead and silver ore are found in porphyry or in quartzite. North of the belt great deposits of copper ore are in the Commercial mine. These carry little or no lead and lie in a limestone (Commercial) above the Jordan and between the Jordan lead-and-copper deposits and the Bingham stock.

In the Last Chance mine in the porphyry of the last Chance stock, a belt of lead-silver ores in fissures extends southwest to the Hooper shaft and lead ores are found in the Mountain Gem mine in limestone included in the porphyry. The Stewart mine, north of the Last Chance, worked gold and copper ore in limestone. In the area north of the Last Chance intrusive and west of the Bingham stock, a great series of deposits included in lenses of limestone lie in quartzite dipping steeply north. These deposits, which have recently been described by R. N. Hunt, include the great deposits of the Utah Metals, Utah Consolidated and

Utah Apex companies and are concentrated along fissures and in fractured masses in the Highland Boy and Yampa limestones and in thinner limestone beds above the Yampa.

Near the great finger of porphyry that extends west from the Bingham stock copper ores predominate. These are most extensively developed in the Utah Consolidated (Highland Boy) and Yampa mines. Farther southwest lead-silver ores were mined in the Utah Metals ground. Lead deposits are found as far west as the Zelnora mine. Very little copper is present in the Utah Metals deposits, although the mine lies along the border of the Last Chance intrusive. Down the dip of the Highland Boy limestone in the Utah Consolidated mine, along the border of the Bingham intrusive, great bodies of copper ore were worked. To the north, down dip in the same beds and farther from the Bingham intrusive are large deposits of lead ore. Copper in large amounts was mined in the Yampa limestone near the Bingham intrusive. In this limestone also lead ore appeared in large amounts down dip and to the north in the Utah Apex ground.

Still further north lead ores with silver, gold, and a little copper were found in the York Phoenix and Petro. North of Petro lead ores are found. In the Fortuna, Congor and Blaine mines northeast of the Bingham stock, ores with lead, silver and some copper are found in fissures in quartzite, or along quartzite and some other rock. In the Fortuna mine lead decreased and copper increased with depth. The Richmond and Dalton and Lark carry lead-silver ores in limestone.

Briefly, the ores of the Bingham district show a concentric arrangement about the Bingham stock: 1, Copper ores are found in the stock (Utah Copper) and at places near its border (Ohio, Commercial, Highland Boy); 2, copper and lead ores are found farther from the stock (Telegraph, Niagara, Old Jordan, Brooklyn, and others); and, 3, lead ores are found farther away (Fortuna, Richmond, Galena, Queen, Last Chance, Utah Metals, Utah Apex, Tiewaukee and others). Gold is generally more closely associated with copper ores and silver with lead ores, but both silver and gold are present in most of the ores of the district.

Tintic, Utah

The Tintic district,[20] Utah (Fig. 14), about 40 m. south of Bingham, is one of the foremost mining centers in the United States, yielding in order of their value (to 1917) silver, lead, gold, copper and zinc. The central

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[20] W. Lindgren, and G. F. Loughlin and V. C. Heikes: Geology and Ore Deposits of the Tintic Mining District, Utah. U. S. Geol. Survey *Prof. Paper* 107 (1919), 1–282.

G. W. Tower, Jr. and G. O. Smith: U. S. Geol. Survey 19*th Ann. Rept.*, Pt. 3 (1898) 601–767.

G. W. Crane: Geology of the Ore Deposits of the Tintic Mining District. *Trans.*, **54** (1916), 342–355.

feature of the district is the Silver City monzonite stock, 1½ m. in diameter, intruded in the Paleozoic sedimentary rocks and Tertiary volcanics. The chief deposits are in limestone and extend northward from the stock along zones of fracturing in limestone.

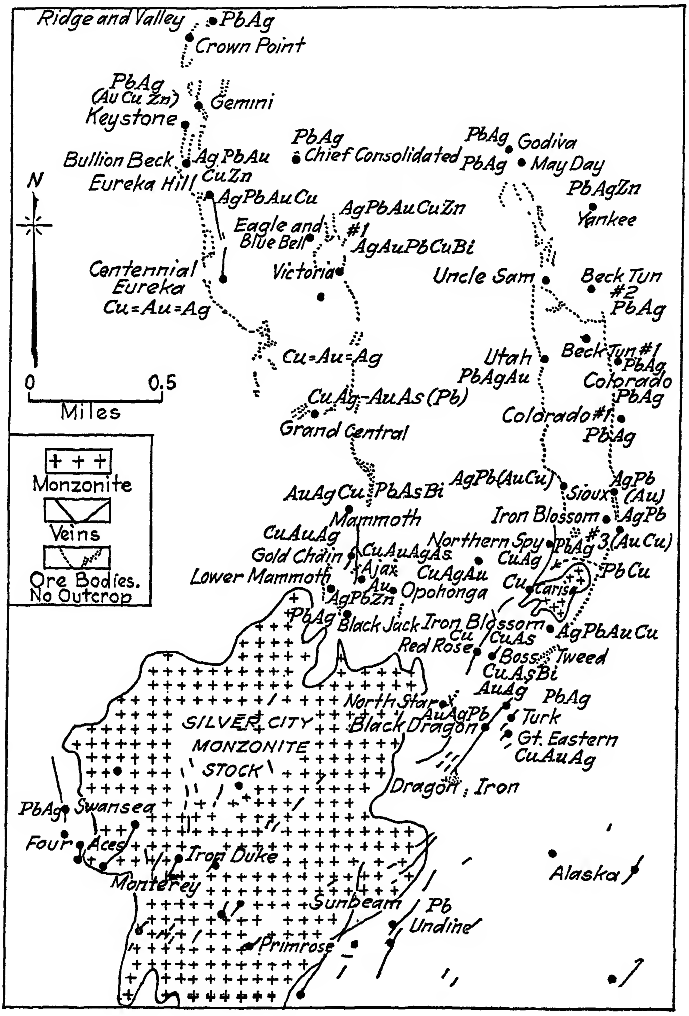

FIG. 14.—SKETCH OF TINTIC, UTAH, SHOWING THE DEPOSITS ECCENTRIC WITH RESPECT TO THE SILVER CITY STOCK. THE EPICENTRUM SEEMS TO BE A BELT EXTENDING NORTHWEST FROM THE CARISA STOCK. THE SERIES IS: 1, COPPER AND ARSENIC; 2, COPPER, LEAD, SILVER; 3, LEAD, SILVER. (DATA FROM LINDGREN, LOUGHLIN, TOWER, SMITH AND CRANE.)

A few deposits in the stock were worked for lead ore. The ore carries much pyrite with galena, enargite, sphalerite and chalcopyrite, the principal values being in lead and silver. The Swansea mine in porphyry just

west of the stock worked a lead-silver vein. The main orebodies lie north of the stock in four great zones of limestone.

Tintic is the only important lead-copper district known to the writer, in which the lead ores occur nearer than copper to the central intrusive. The district offers an example of either an eccentric series or a reversed series. As copper appears with depth in some of the veins that carry only lead in the upper levels,[21] and as none of the copper ores pass into lead ores below, it appears probable that the metallization of the district is not reversed, but is eccentric with respect to the Silver City stock. The copper ores with associated arsenic and locally bismuth are normally the deepest seated types of ores of the lodes. They lie in a belt off center from the Silver City intrusive extending from the Carisa intrusive northwest to the Centennial Eureka mine, and include the copper and copper-lead deposits of the Iron Blossom, Carisa, Northern Spy, Gold Chain, Mammoth, Grand Central, and Centennial Eureka mines.

North of these deposits in each of the four ore zones, copper decreases and lead-silver ores increase. To the south of this belt of copper ores, lead increases in the Mammoth zone in limestone and in the lode deposits in the igneous rocks. In view of the relations in certain other districts, it seems not improbable that a crest of an igneous mass, mainly concealed extends, northwest from the Carisa intrusive to the Centennial-Eureka mine, and that the solutions rose from this crest, spreading out to the north and south. This theory receives some support from the close spacing of dikes at the ends of the copper belt. Gold ores in general are closely associated with those of copper. The copper ores generally carry silver and all of the lead ores carry silver.

The main deposits in the district lie in the four zones of fissuring that extend northward from the Silver City intrusive. The deposits are commonly in steep veins at the south ends of those zones. Toward the north, bedding plane deposits predominate. They are generally at intersections of fissures and favorable beds. Cylinders or ribbons of ore are common and where the beds are nearly flat-lying, as in the east part of mineralized area, the ribbons are almost horizontal. Where the beds are nearly on edge, as in some of the great mines near Eureka, the ore shoots are more nearly vertical.

The Iron Blossom ore zone at the east edge of the district extends northeast across the limestone beds from the main monzonitic mass. At the south end it is a well defined fissure vein, but near Iron Blossom No. 3 the deposit becomes a flat thick ribbon of ore 20 to 70 ft. wide, 20 to 60 ft. high and 5000 ft. long. This part of the deposit is at the crossing of the fissured zone of the Iron Blossom with a favorable bed of limestone, which

[21] W. Lindgren and G. F. Loughlin: The Ore Deposits of Utah—The East Tintic Mountains—U. S. Geol. Survey *Prof. Paper* 111 (1920) 414.

is replaced. The fissure follows the axis of a syncline and its intersection with the beds is nearly flat. The limestone is intruded by many dikes near the Carisa stock, and there is some metamorphism near the dikes. The ores carry silver, lead and gold; near the Carisa stock and to the south of it toward the Silver City stock, in and south of the Iron Blossom No. 1 mine, the ore carries also copper. In the Iron Blossom No. 3 and to the north, the lode carries more lead than the part of the lode to the south and contains little copper. Still farther north in the Colorado and Sioux, lead increases to 45 per cent., silver to 45 oz. per ton, and gold decreases.

The Godiva zone lies nearly parallel to and about one-fourth mile west of the Iron Blossom zone. It is developed on strike 12,000 ft. South of the Northern Spy the ores carry mainly copper, gold and silver. North of the Northern Spy for 7000 ft. they carry silver-lead ores. In this zone also the copper ores lie near and south of the Carisa intrusive, whereas the lead-silver ores extend as a flat-lying ribbon far to the north of it. In the lower levels of the zone south of the Carisa, the veins carry arsenical copper ore.

The Mammoth zone, about one-half mile to three-fourths mile west of the Godiva zone, is about 10,000 ft. long and consists of a series of closely spaced veins and bedding-plane deposits in limestone in which the ore shoots commonly pitch steeply. At the south end of the zone, lead-silver ores with some gold are found in the Black Jack and Lower Mammoth mines. The Ajax to the north yielded chiefly gold; the Gold Chain north of the Ajax, and the Mammoth farther north yielded copper, gold, silver, arsenic and bismuth with very little lead. North of the Grand Central mine, lead increases and copper decreases and the deposits at the north end of the zone in the Chief Consolidated mine are lead-silver ores with little copper.

The Gemini zone lies west of the Mammoth zone and extends northward 8000 ft. The ore beds occur as pod-like masses along four almost vertical fracture zones which coincide essentially with the northward striking dolomite beds. At its south end this zone joins the Mammoth zone in the Grand Central mine. The Centennial-Eureka mine at the south end of the Gemini zone is the most productive mine of the Tintic district. The orebodies carry copper, gold and silver, the value of the metals being about equal. Only a little lead is present. In the Eureka Hill mine to the north, silver-lead ore predominates, although some copper and gold are mined. Farther north in the Eureka Hill and Bullion Beck and Gemini mines, lead and silver increase and copper decreases and in the Ridge and Valley mine near the north end of the ore zone, the ore carries lead and silver essentially without copper. Secondary zinc carbonate ores are mined in the Gemini zone near the lead-silver stopes.

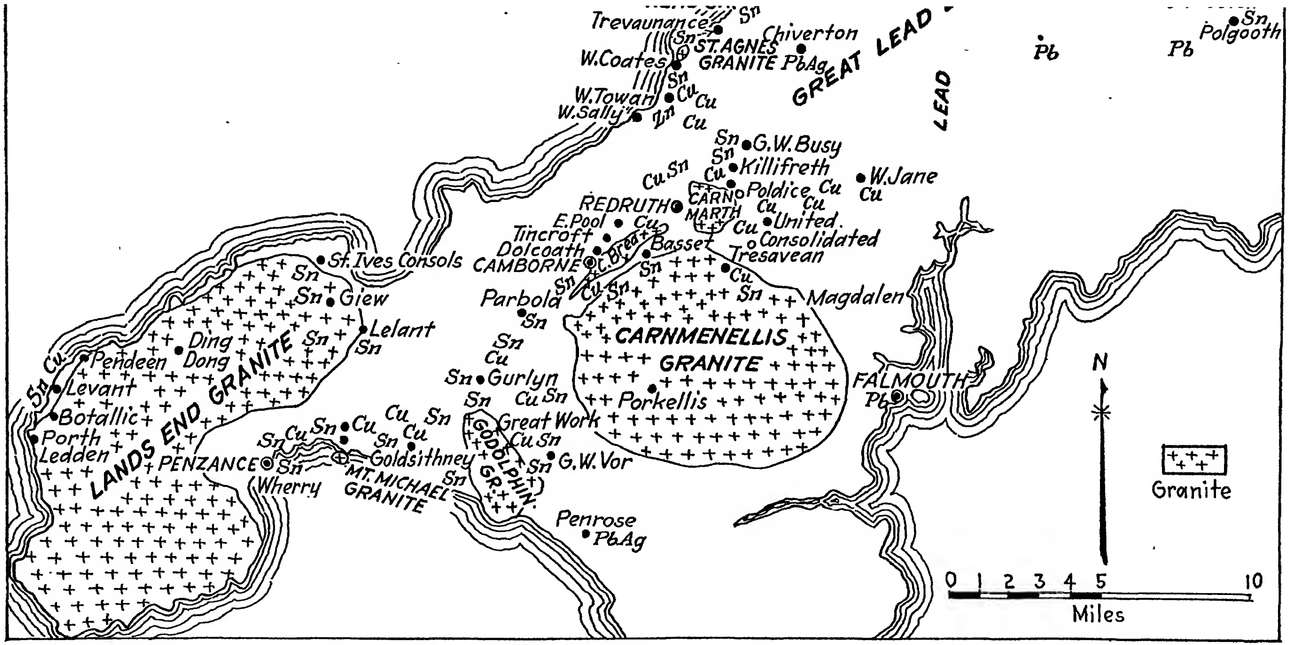

Fig. 15.—Map of west end of Cornwall, England, showing the series: tin, tungsten, arsenopyrite, copper, zinc, lead, silver. (Map from Geological Survey of Great Britain. Mine locations from Collins, Dewey, Reed, MacAlister, Hill, and others.)

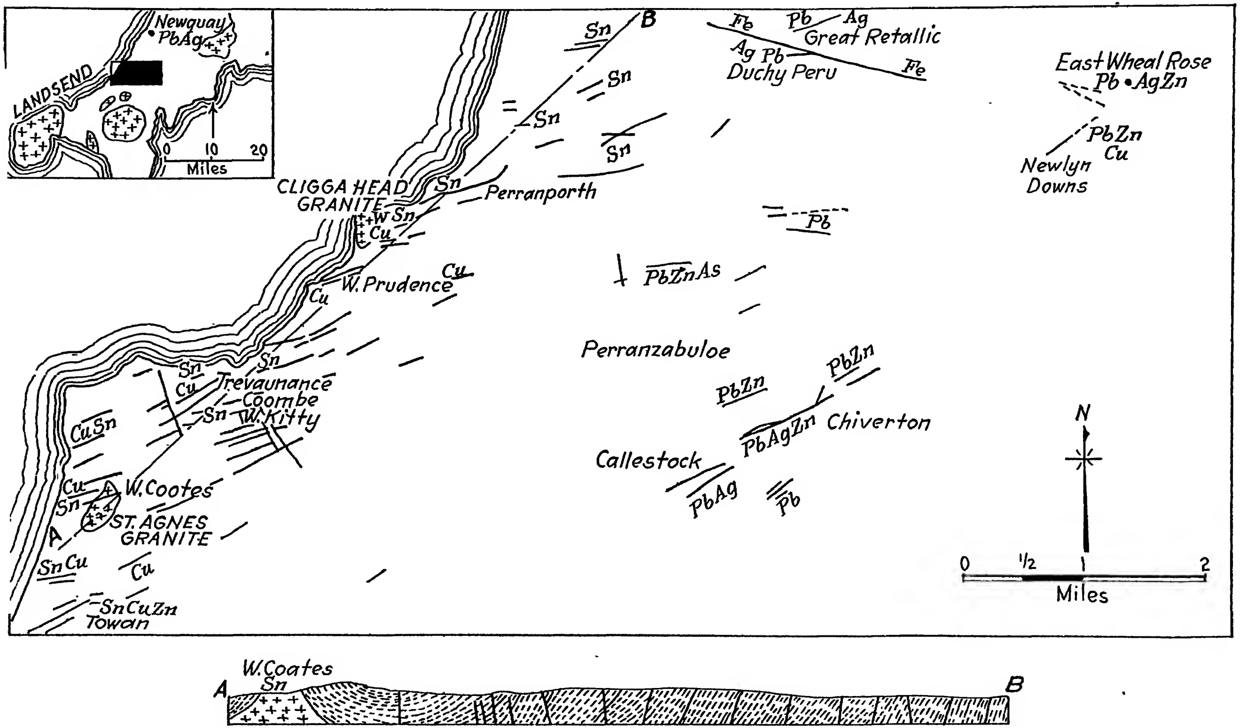

Fig. 16.—Map of St. Agnes and Cligga Head mining area, Cornwall, England, with section on a northeast line along coast oriented on AB. The line of section is the zone of tin, tungsten and copper ores. To the east is one with lead, silver and some zinc. (Based on map by Reid, Scrivenor, and MacAlister.)

St. Agnes and Cligga Head, Cornwall, England

In certain mining districts where two or more cupolas are closely spaced and each shows concentric zones of metallization, these zones join between the cupolas. The epicentrum lies along a line joining and extended beyond the two cupolas. Examples of this arrangement are the deposits near St. Agnes and Cligga Head in Cornwall, Fig. 15, and the deposits near Tavistock in Devon and Cornwall.

The St. Agnes and Cligga Head[22] granites are small bodies exposed along the coast nearly due north of Camborne and Redruth (Fig. 16). They cut through Devonian sedimentary rocks and probably represent high points on a comb or ridge of an underlying batholith. Tin veins and a few veins with copper ore are found in and near the granite and the tin-bearing zones extend far to the northeast and southwest of the exposures of granite.

South of St. Agnes, zinc ores appear in the Towan mine. To the southeast, east, and north of the tin belt great deposits of lead-silver ores have been mined. The chief centers are at Great Retallic, Duchy Peru, Callestock, and Chiverton. At St. Endelleon still farther north beyond the area shown on Fig. 15, are deposits of antimony and lead. The series passing outward from the granite is the one shown at many other places in Cornwall, namely, tin, copper and a little tungsten ore near the granite outcrops and along a line joining them, and, zinc, lead and silver farther east.

Tavistock, Devon and Cornwall

The Tavistock[23] mining region, which lies about midway between the great granite masses of Dartmoor and Bodmin Moor, centers around two small outcrops of granite (Fig. 17). One of these is the Gunnislake, or Hingston Down[24] mass, 4 m. southwest of Tavistock; the other, the Kit Hill mass, is 2 m. west of Hingston Down. The epicentrum is believed to include the area in and around the two intrusives and extending east and west from them. In this area deposits of tin, copper, tungsten and arsenopyrite are extensively developed. On either side are deposits of lead, and silver deposits have been mined at many places.

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[22] C. Reid and J. B. Scrivenor: The Geology of the Country Near Newquay. Mem. Geol. Survey Great Britain, Explanation of *Sheet* 346 (1906), 1–131.

[23] C. Reid, G. Barrow, R. L. Sherlock, D. A. MacAlister, and H. Dewey: The Geology of the Country Around Tavistock and Launceton. Mem. Geol. Survey Great Britain, *Sheet* 337 (1911), 1–144.

[24] Called Gunnislake mass in some of the earlier reports of the British Geological Survey, and Hingston Down mass in later ones.

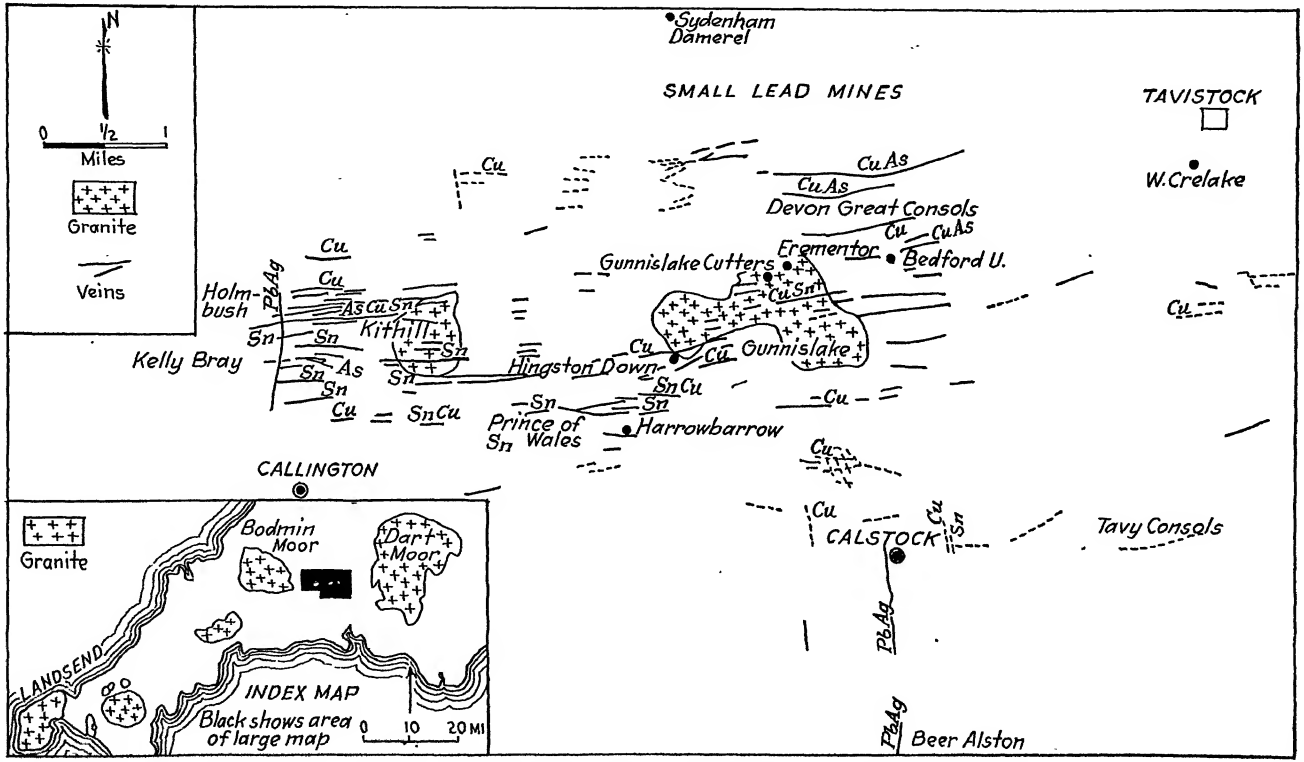

FIG. 17.—MAP OF TAVISTOCK DISTRICT, CORNWALL AND DEVON, SHOWING THE EPICENTRAL AREA WITH TIN, ARSENIC AND COPPER, WHICH IS AN ELONGATED BELT INCLUDING THE TWO GRANITE INTRUSIVES. ON EITHER SIDE OF IT ARE DEPOSITS OF LEAD-SILVER ORE. (BASED ON MAPS BY REID, BARROW, USHER, SHERLOCK, MACALISTER, AND DEWEY.)

GROUP 3: EPIBATHOLITHIC DEPOSITS

The term epibatholithic is appropriate because nearly all of the deposits of this group are on and near the edge of the intrusive in the invading rock, or outside of the intrusive in the invaded rock. The line of division between this group and the acrobatholithic group, the deposits in and near cupolas, (group 2), is somewhat arbitrary. The rocks of the cupolas include both the granitic rocks and the deeper seated porphyries. All of the intrusives around which the deposits of group 3 are centered are granular in part or altogether.

The cupolas range from less than 1 m. to 3 or 4 m. wide. In general, the intrusives around which the deposits of group 3 center are more than 3 m. wide and have essentially barren centers. That is almost invariably true of the larger masses, but even a few of the larger masses contain deposits near their centers. Most of the latter are probably flat-roofed. Where the moderately large intrusives are closely spaced the type grades into group 4, the embatholithic group. With depth, however, the deposits of the metals other than gold decrease in number and value.

Lode deposits, as already noted, are features of the roofs of batholiths. The centers of the larger intrusives are generally barren of valuable deposits except in and near roof pendants. Moreover there are few, if any, intrusives with rims that are uniformly metallized. That is true probably because fractures are not uniformly present to afford suitable channels for the passage of solutions. In some districts the deposits of this group are localized near places where the outcrop bulges outward or forms a finger extending into the invaded rock, or where a smaller satellitic cupola is exposed outside of the larger outcrop of the intrusive. The outlying cupolas often are metallized, as other cupolas, with series of lodes showing zonal arrangements. The Carn Marth and Carn Brea granite masses, around which are centered many of the greatest mines of Cornwall, lie near and north of the Carnmenellis granite. That they are cupolas on the larger underlying massive is established by underground workings. Examples of satellitic deposits to which fingers point are Horn Silver and Carbonate mines of Frisco, Utah.

Many of the fingers of batholiths at a certain earlier stage of erosion would probably be exposed as smaller satellitic cupolas. The satellitic cupolas and the regions in and near fingers of batholiths are favorable situations for the same reason. They are relatively high places on the roof and, where fractures are supplied, the rising solutions tend to move to such places just as they tend to move to the high cupolas of group 2. Lower down on the batholiths, this feature of their metallization tends to disappear and in group 4 the satellitic cupola is more commonly barren.

Camborne, Cornwall, England

The Cornwall region, England, which embraces essentially the county of Cornwall and a small area in west Devon, is about 100 m. long and 10 to 30 m. wide, and is composed mainly of Paleozoic sedimentary rocks,[25] intruded by granite, probably of Permian age. From west to east the larger granite masses (Fig. 15) are: Land's End, Carnmenellis, St. Austell, Bodmin Moor, and Dartmoor. Smaller granite masses include Godolphin, M. Michael, Carn Brea, Carn Marth, St. Agnes, Cligga Head, Kit Hill, Hingston Down and others. There are also many elvan or quartz porphyry dikes not shown on Figs. 15, 16, 17 and 18.

The deposits[26] are essentially all veins. They contain tin, tungsten, bismuth, arsenic, copper, zinc, lead, silver and antimony, named approximately in order of their distance from the granite.

The veins are mainly in and around the smaller stocks and in granite at the edges of the larger intrusives and around the larger intrusives. The centers of the larger intrusives carry few deposits. The largest granite mass, the Dartmoor granite, is essentially barren. Around the Land's End granite (Fig. 15) and in its rim are found the Botallic, Levant, Pendeen, St. Ives, and other mines. These carry tin, but copper appears in some of the veins as they are followed away from the granite. The bulk of the metallization around the Carnmenellis granite is near the small satellitic cupolas. In and around these intrusives (Fig. 18) veins are closely spaced; this center is the most highly metallized part of the country and one of the richest mining centers of the world.

Large deposits are found also around the Godolphin satellitic granite (Fig. 15) and northward around the St. Agnes and Cligga Head masses already mentioned. The series, passing away from the granite, as already

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[25] H. T. De la Beche: Ordnance Geol. Survey, London (1839), 1–648.

W. S. Henwood. Royal Geol. Soc. Cornwall, **5** (1843) 1–386.

J. H. Collins. *Trans.*, **14** (1911).

J. B. Hill, D. A. MacAlister, and J. S. Flett. *Mem.* Geol. Survey Great Britain, Explanation of *Sheet* 352 (1906), 1–334.

C. Reid and J. S. Flett. *Mem.* Geol. Survey Great Britain, Explanation *Sheets* 351 and 358, (1907), 1–158.

C. Reid, J. B. Scrivenor: *Sheet* 346 (1906), 1–131.

H. Dewey: Tungsten and Manganese. *Mem.* Geol. Survey Great Britain, **1** (1915), 1–14.

H. Dewey: Arsenic and Antimony Ores. *Mem.* Geol. Survey Great Britain No. 15, 1920; No. 21 (Lead, silver and zinc), 1921; Sheet 337 (Tavistock and Launceton) (1911), 1–144.

R. H. Rastall: Metallogenetic Zones. *Econ. Geol.*, **18** (1923), 105–121.

[26] The bulk of the production is tin, copper, lead, and silver. D. A. MacAlister (Geological Aspects of the Lodes of Cornwall. *Econ. Geol.,* **3** (1908), 363–380) estimates the tin production to 1905 to be about 2,015,890 tons of metal, and the copper production between 1501 and 1905 to be 903,350 tons of metal. About 63,600 tons of lead and 2,930,000 oz. silver were produced between 1848 and 1884.

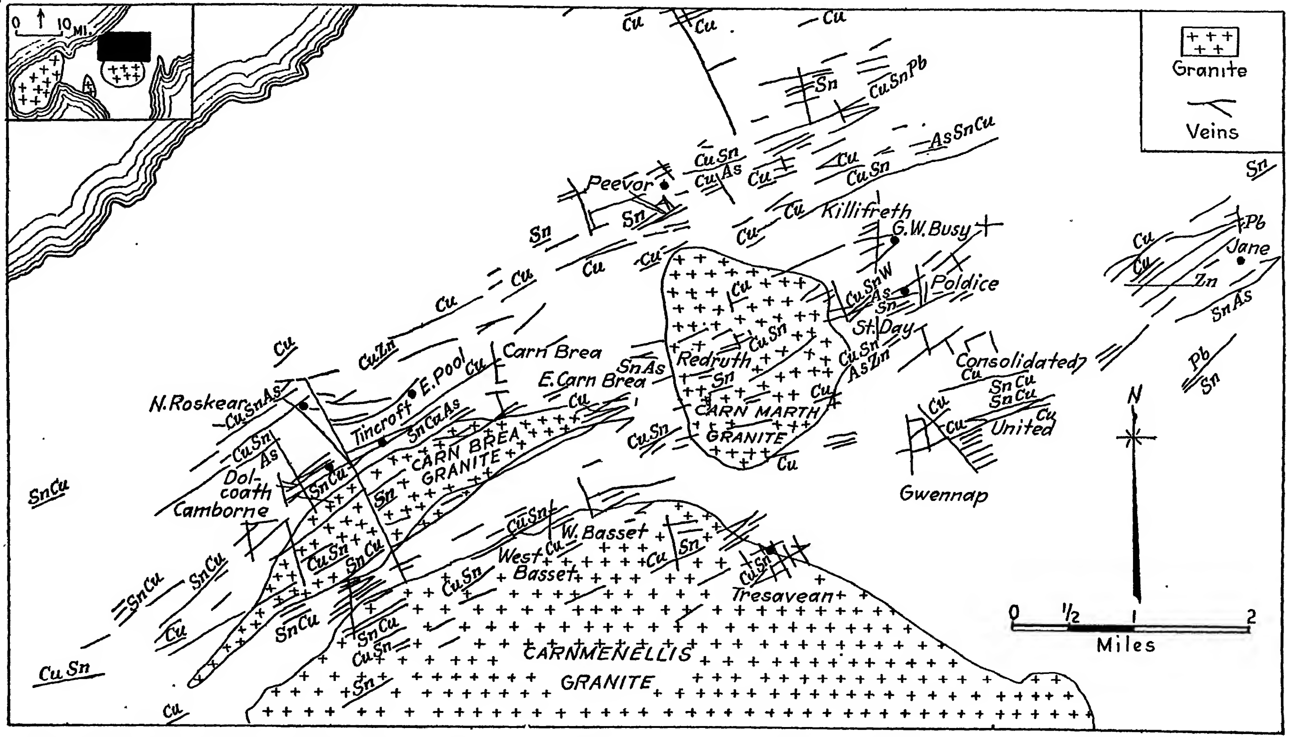

FIG. 18.—MAP OF AREA NEAR REDRUTH AND CAMBORNE, CORNWALL, ENGLAND, THE MOST PRODUCTIVE PART OF CORNWALL, SHOWING THE DEPOSITS CLUSTERING AROUND TWO SMALL GRANITIC BODIES, CARN BREA AND CARN MARTH, WHICH ARE SATELLITES OF THE LARGER GRANITE MASS, CARNMENELLIS. THE SERIES, PASSING AWAY FROM THE GRANITE, IS TIN, TUNGSTEN, ARSENIC, COPPER, ZINC, LEAD, SILVER. (DATA FROM HILL, MACALISTER, AND FLETT.)

noted, is: tin, tungsten, arsenic; copper, sphalerite, lead, silver and antimony. This series is established by the positions of the veins on the surface, by changes in single veins that extend outward from the granite, and by changes downward in single veins.

This is a classic area for the study of successive zones; it was here that they were first recognized and prospecting was carried on with the standard changes in mind. The zones were recorded by Hill, Reid, MacAlister, and others in the various papers cited. They are best summarized in a series of economic papers by Dewey and by recent general papers by Rastall, Dewey and Davison. Because of the number of intrusives and the variety of metals present, Cornwall is probably the

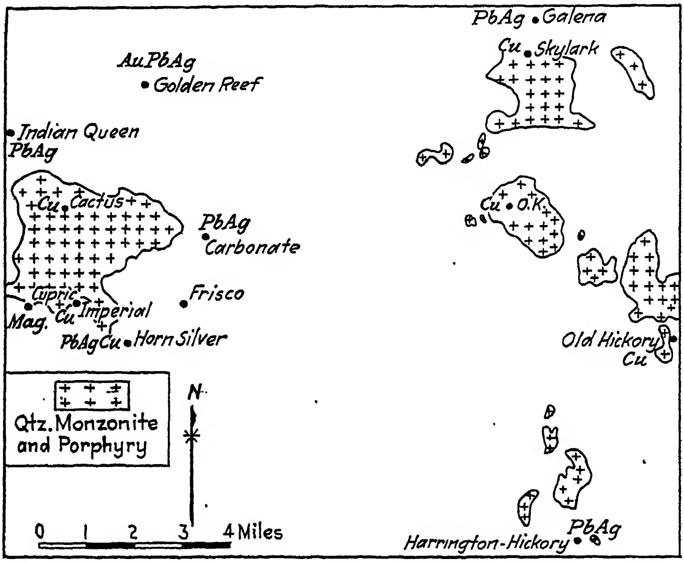

FIG. 19.—SKETCH OF PART OF SAN FRANCISCO REGION, UTAH, SHOWING COPPER ORES IN AND NEAR THE INTRUSIVES AND LEAD SILVER ORES FARTHER OUT. THE HORN SILVER AND CARBONATE MINES ARE NEAR FINGERS OF THE INTRUSIVE. (DATA FROM BUTLER.)

most instructive area known for the study of zones. Moreover, in western France granites of the same age are found. There erosion has gone deeper and although approximately the same metals are present, zones are not recognized and gold appears in quantities economically valuable.

San Francisco Region, Utah

In the San Francisco[27] district near Newhouse, Utah, Paleozoic sedimentary rocks and also Tertiary lavas are intruded by a quartz monzonite stock about 3 m. in diameter (Fig. 19). The Cactus mine in

[27] B. S. Butler: Geology and Ore Deposits of the San Francisco and Adjacent Districts, Utah. U. S. Geol. Survey *Prof. Paper* 80 (1913), 1–212.

the monzonite is an irregular, wide, short lode or chimney, of copper ore with chalcopyrite, pyrite, hematite, tourmaline and anhydrite, barite and siderite. The ore fills spaces between fragments of the quartz monzonite. In limestones near the south border of the intrusive, magnetite ore has been mined at the Cupric mine and some copper ore is found at the Imperial to the east.

At the Horn Silver mine, 3 m. southeast of the Cactus mine, lead-silver ore lies in a fault zone between limestone and Tertiary lava. The deposit which is one of the largest in the district is near the end of a finger of the intrusive which is said to be encountered on level 11. Near the surface silver-lead ore predominates, but in depth secondary copper sulfide ore and zinc sulfide ore become locally prominent.

The lead-silver ores of the Carbonate silver-lead mine, 3 m. east of the Cactus, are in lavas also near the end of a finger of the monzonite. At the Indian Queen mine, 2 m. northwest of the Cactus, lead-silver ores in limestone have been mined. Gold ores with silver, lead have been mined at the Golden Reef mine 3 m. northeast of the intrusive mass.

Northwestern Tasmania

In northwestern Tasmania[28] schists and early Paleozoic sedimentary rocks are intruded by serpentine and by granitic rocks (Fig. 20). The latter are younger than the serpentine but both are probably Paleozoic. Associated with the serpentine are placer deposits of osmiridium that constitute the world's chief supply of point metal for pens. Along the west coast tin deposits are found at several places, all associated with granite porphyry, except some small deposits at Mt. Balfour where no granitic rock is exposed.

At Mt. Heemskirk (Fig. 21) cassiterite ore with some tungsten and bismuth are found in the granite; some tin ore is found also in the invaded rock. East of Mt. Heemskirk are some small deposits of chalcopyrite, sphalerite and pyrite, with some cassiterite and stannite. Farther east are the silver-lead deposits of Zeehan and Dundas. Northeast of Zeehan, some tin ore is found at Renison Bell near a great dike of porphyry. Lead, zinc and silver deposits are found between Tullah and Mt. Read and pyrite is mined at Chester.

To the north in the Merideth range, tin ores as placers are found near Parsons Hood, Castra River and Mt. Cleveland. Northeast of

[28] G. A. Waller: Report of the Secretary for Mines (Tasmania) 1902–1903 (1903), 11–24.

R. M. Johnston: Geology of Tasmania (1888) 1–408.

W. H. Twelvetrees and L. K. Ward. Geol. Survey Tasmania *Bull.* 8 (1910), 1–165.

H. Couder: Geol. Survey Tasmania, *Bull.* 26 (1918), 1–96.

L. K. Ward: Report of the Secretary for Mines of Tasmania for 1909 (1910).

Mt. Cleveland lead-silver ores have been mined at Magnet and still farther to the northeast great deposits of tin ore in porphyry have been mined at

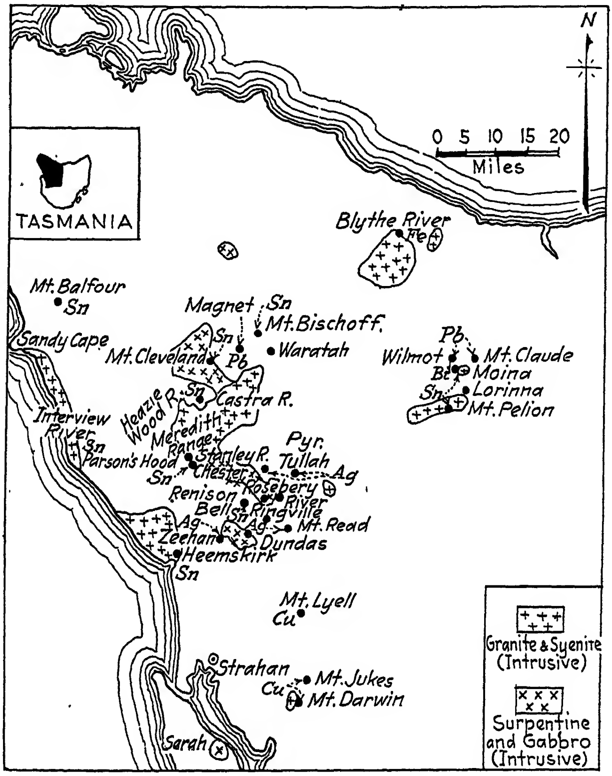

FIG. 20.—SKETCH OF MAIN METALLIZED AREA OF NORTHWEST TASMANIA, SHOWING INTRUSIVE SERPENTINE AND LATER INTRUSIVE GRANITE. OSMIRIDIUM AND CHROMIUM ARE FOUND ASSOCIATED WITH THE SERPENTINE. TIN ORE IS IN AND NEAR GRANITE AND LEAD ORE FARTHER AWAY. THIS SERIES IS SHOWN AT HEEMSKIRK, PARSON'S HOOD, MOUNT BISCHOFF, AND MOINA. (DATA FROM WALLER, TWELVETREES, AND JOHNSTON.)

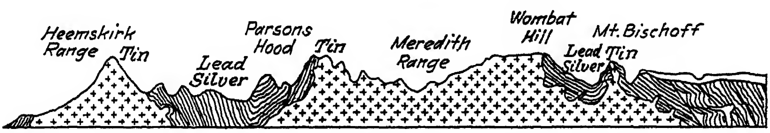

FIG. 21.—GENERALIZED SECTION ACROSS MAIN METALLIZED AREA OF NORTHWEST TASMANIA (SEE FIG. 20), SHOWING THE RELATION OF TIN DEPOSITS AND LEAD-SILVER DEPOSITS TO GRANITIC INTRUSIVES. THE LEAD SILVER DEPOSITS BETWEEN HEEMSKIRK AND PARSON'S HOOD LIE SOUTH OF THE LINE OF SECTION AND THOSE BETWEEN MERIDETH RANGE AND MOUNT BISCHOFF ARE NORTHWEST OF IT. (SECTION BASED ON ONE FROM JOHNSTON.)

Mt. Bischoff. One of the latter deposits is noteworthy in that lead-silver ore appears in depth in one of the tin veins, but according to Weston-

Dunn,[29] this ore is later in age than the tin ores. At Moina east of Mt. Bischoff, tin, tungsten and bismuth ores are found in and near the border of a granitic intrusive 2 or 3 m. in diameter. Lead ores occur north of the tin deposits.

Group 4: Embatholithic Deposits

The deposits of the embatholithic group lie among closely spaced granite masses, which are the high parts of intrusives. Further erosion would expose the invaded rocks as roof pendants surrounded by the invading rocks. In such surroundings gold deposits nearly everywhere predominate, but there are also deposits of copper, zinc and other metals. The granitic rocks are generally barren except near contacts. In the central parts of the intruded areas are few if any well defined zones, but near the margins of the areas where the granitic outcrops decrease in number or in size, or where they disappear altogether, the deposits are likely to appear in broad zones. These are not everywhere well defined, but the zones may locally be recognized when the deposits are plotted on relatively small-scale maps showing large areas. The embatholithic type of metallization is shown in southwestern Oregon, in northwestern California, in the Sierra Nevada area of California, in the southern Appalachians, in the main gold-bearing regions of Siberia and of Egypt and in Victoria.

In southwestern Oregon, (Fig. 22) gold lodes are found among the granitic outcrops in the invaded rocks and a few deposits occur in the invading rocks near their borders. Locally mercury deposits are developed, but these are probably of much later age than the gold deposits. The same type of metallization extends southward into the Klamath Mountain region of California. In this area, gold deposits predominate except in the Redding district, Shasta County, where large copper and zinc deposits are developed. This district (Fig. 10) is near the edge of the main gold-bearing embatholithic area and the deposits should probably be regarded as an acrobatholithic group at the edge of the larger area of embatholithic deposits. The series is probably 1, copper, 2, gold.

In the Sierra Nevada region also the embatholithic type of metallization is shown. The chief deposits carry gold and copper ores. Nearly all of them are in the invaded rocks, although a few are found in the invading granodiorite near the contacts. Zones are not well shown except on the western border of the region where the foothill copper belt lies parallel to the gold deposits of the Mother Lode and farther from the main Sierra batholith.

In California south of the area shown in Fig. 22, the Sierra Nevada batholith contains many roof pendants with characteristic gold deposits

[29] J. G. Weston-Dunn. *Econ. Geol.*, 17 (1922), 153–193.

in the invaded rocks. To the east of the main intrusive, however, there are many smaller granite intrusives with associated deposits of the cupola or acrobatholithic type.

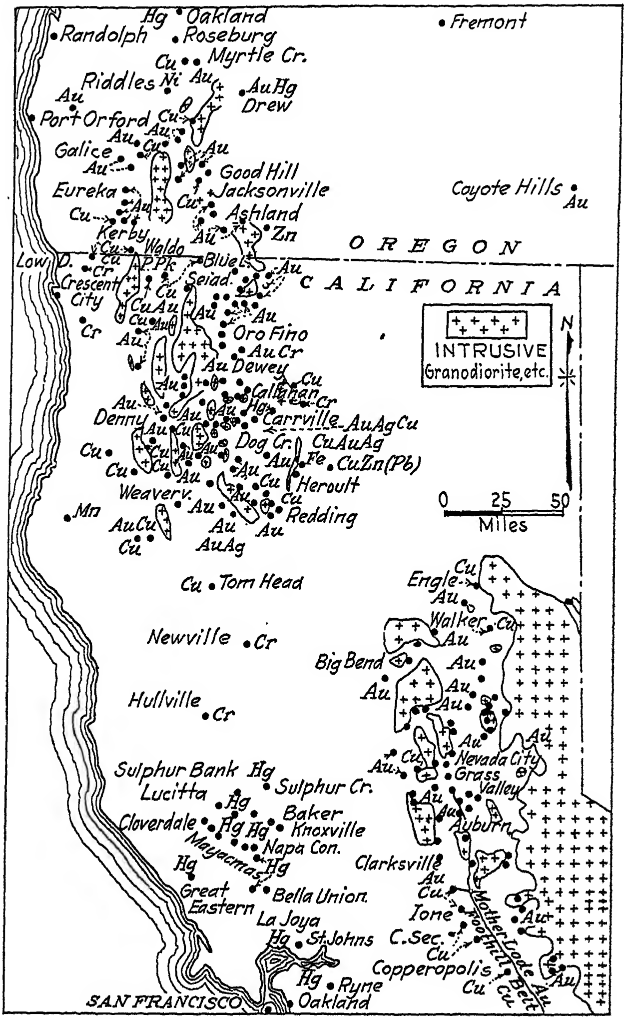

FIG. 22.—MAP OF SOUTHWEST OREGON AND NORTHERN CALIFORNIA, SHOWING THE RELATION OF THE CHIEF MINERAL DEPOSITS TO GRANITIC INTRUSIVES (EMBATHOLITHIC TYPE OF METALLIZATION). GOLD DEPOSITS GREATLY PREDOMINATE, ALTHOUGH VALUABLE COPPER DEPOSITS OCCUR. DEPOSITS ARE IN THE INVADED ROCKS, IN SMALL BODIES OF THE INVADING ROCKS, OR AT THE EDGES OF LARGER BODIES OF THE INVADING ROCKS. (DATA FROM DILLER, KAY, WINCHELL, MACDONALD, BRADLEY, GRATON, HERSHEY, LINDGREN, RANSOME, TURNER, AND OTHERS.)

The Southern Appalachian Mountains exhibit a type of metallization which in certain respects is similar to that of the Sierra Nevada. Gold

deposits prevail among closely spaced batholiths, but west of the area of closely spaced intrusions other metals appear in force. These are arranged in zones as follows: 1, gold and some tin; 2, copper; 3, manganese; 4, zinc; 5, zinc and lead; 6, barite. Although a few deposits are out of their normal positions, the general trend of the metallization is clear. The occurrence of these deposits approximately in their normal order, all in rocks as old as Carboniferous in and around Carboniferous granite, strongly indicates that they are related genetically to the Carboniferous granite. In the southern Appalachians, as in California, the larger granitic masses are barren except near contacts.

In Egypt, in the main gold-bearing regions of Siberia and in Victoria, there are great areas of embatholithic deposits. Except in the Altai of Siberia, gold greatly predominates. Although erosion has exposed large areas of the granitic batholiths in all of these regions, it has not gone far enough to develop the roof pendant stage. Practically all of the gold deposits are in the invaded rocks, although a few are in the intrusives near contacts and certain placer deposits which appear to be of local origin have accumulated within the granite along gold-bearing streams in the Minoussinsk district in Siberia.

Group 5: Endobatholithic Deposits

Probably all large granitic batholiths have undulating roofs. At a certain stage in erosion great remnants of the roofs remain surrounded by the granitic intrusive and projecting into the intrusive. The frame of the endobatholithic area is the intruding rock, whereas the frame of the embatholithic area is the intruded rock. The deposits in the areas of roof pendants are nearly all in the roof pendants or in the invading rocks near the roof pendants. Gold deposits greatly predominate, but a few valuable copper deposits and a small number of lead, silver, zinc, and other metals are found. This is probably the most productive type of auriferous metallization. It includes the chief gold deposits of Ontario, western Quebec and eastern Manitoba; the gold deposits of southern Rhodesia and those of Western Australia. It is highly probable that the great bankets of the Rand were derived from deposits of this type in southern Rhodesia and other similar deposits near by.

Fig. 23, a map of the chief auriferous area of Western Australia, illustrates this type of metallization. The maps of Rhodesia and Canada are very closely similar in that they show the same geological situation and a similar distribution of deposits. In all of these districts gold greatly predominates.

In southern California and in the northern part of Lower California the metallization is similar, although the intrusives are much smaller than in the other districts named.

Group 6: Hypobatholithic Deposits

The deposits of the hypobatholithic group are in granitic masses which are very deeply eroded. Over such masses the roof has been

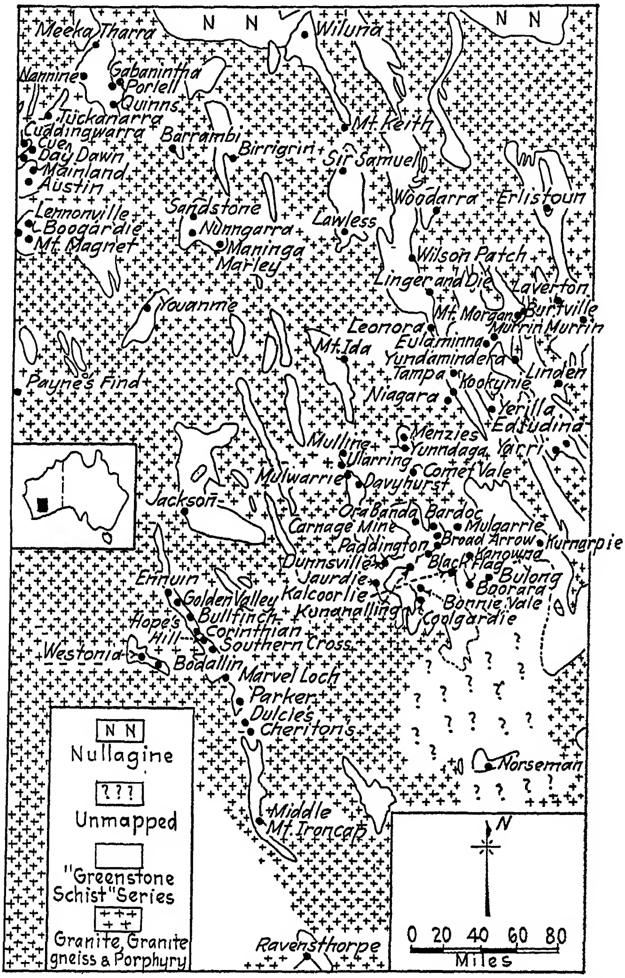

Fig. 23.—Map of the most productive part of Western Australia, showing the mining districts (by dots), which contain deposits of gold and small amounts of other metals. About 97 per cent. of the total production is gold. The deposits are all in or near the borders of the roof pendants of invaded rocks; the invading rocks are generally barren. (Data from Maitland, Clark, Talbot, Saint-Smith, Montgomery, Simpson, Feldtman, Honman, Blatchford, Gibson, Rickard, Jackson, Jutson, Woodward, Campbell, and others.)

removed so that few roof pendants remain. The very deeply eroded granitic masses in the main are barren of large valuable deposits, although

they commonly contain veins of quartz and pyrite and in some of them gold veins, generally of low grade, are found. One of the largest granitic masses known is that which occupies the great Kola peninsula between the White Sea and the Arctic Ocean and extends southward to Lake Onega. It covers a large part of northwestern Russia and northern Finland, in all an area of over 100,000 sq. m. This mass seems to be essentially barren except around its southern and western margins where it is intruded by later granites.

The southern part of the granitic area of the great Canadian shield, which seems to contain more and larger roof pendants than the northern part, carries also more valuable lodes of the metals.

Where the deeply eroded granitic masses are intruded by later rocks, they may, of course, contain lodes of a metallogenic epoch that is later than that of the older granite. It is, nevertheless, significant that such large areas of the deeply eroded granite are barren. No other areas of barren igneous rocks of equal size are known except perhaps certain extensive areas of lava flows of late age.

Resumé

The great majority of metalliferous lodes are related to batholiths. These are great masses of igneous rocks that become broader with depth. They are probably intruded as masses of intermediate or more basic composition in the main, but differentiate through the series: diorite, quartz diorite, quartz monzonite, grandodiorite, granite. The lode ores are expressed chiefly after the batholith has differentiated through the diorite stage and most lode deposits are associated generically with rocks more acid than quartz diorite.

Lode ores that are associated with the batholiths may be divided in six groups depending upon their positions with respect to the parent intrusives (Figs. 1 and 2). In the first group, the parent batholiths are concealed, but there are sound reasons for believing that they exist below the deposits. In the second group, the summit of the batholith is exposed by erosion. This group is highly productive. In the third group, erosion has gone deeper, exposing larger masses, surrounded by the invaded rocks. This group also is highly productive, but as a rule the interiors of the granite masses are barren. The fourth group is where erosion has gone deeper, revealing more of the granite. The intrusions are larger and more closely spaced and the deposits are nearly all in the invaded rocks in small marginal cupolas and in the larger masses of the invading rocks near their margins. In the fifth group, erosion has gone still deeper, exposing great areas of the invading rocks with roof pendants of the invaded rock surrounded by the intrusive. In group six, erosion has removed even the roof pendants.

The deposits are greatly concentrated in the rim and out from the rim of the batholith. Fig. 3 shows the relative amounts of metals found in the six situations named. The most striking feature of that figure is the concentration of gold in the fourth and fifth groups. The sixth group is by far the poorest one.

A zonal arrangement of the metals is commonly found in group 1, but more often it is not evident. This arrangement is very common in groups 2 and 3, and the metals or groups of metals are generally arranged in the same order. This order is in accordance with vertical zones noted in a normal vein system (Fig. 4). Zones are less clearly shown in group 4 and gold and copper are often reversed. No zones are discovered in groups 5 or 6.

The conclusions above stated are based on the relations shown on about three hundred maps, nineteen of which are presented here. Most of these show deposits of the second and third groups. Maps of areas showing other deposits of groups 1, 4, 5 and 6 will be shown in another paper.

Acknowledgments

It is a pleasure to acknowledge the many courtesies extended to the writer by Henry Dewey of the Geological Survey of Great Britain and by E. H. Davison of Camborne, in connection with a brief visit to Cornwall and a study of its literature in the offices of the British Survey in London.

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Magmas, Dikes and Veins

By Waldemar Lindgren,* Cambridge, Mass.

(New York Meeting, February, 1926)

Introduction

No one would maintain that all ore deposits or all deposits of useful minerals have been formed by the same processes. Generally they have originated by special processes of concentration but these may be of many different kinds. Some deposits have their origin in chemical or biochemical processes of concentration in rivers, lakes or seas, others developed on land by means of chemical reactions of surface waters on various forms of rocks. Certain insoluble substances remained behind; others were dissolved and precipitated in favorable places. This action proceeded in the realm of the oxidizing surface waters or in the deeper zones to which the waters from the surface could find their way. Examples of the latter kind may be found in the extensive copper and vanadium deposits of some sedimentary beds or in the lead and zinc deposits of certain broad limestone strata. In these instances the evidence for any other mode of origin would seem to be very weak indeed.[1] Nor does it seem possible to deny that circulating waters may form deposits of the more common metals along fissures after having dissolved many constituents from paths traversed.

Another very large class has been formed by magnetic processes of concentration.

The last 40 years have been a period of intense activity in the investigation and description of a vast number of deposits; all modern methods have been employed and the result is an accumulation of data, difficult or impossible to master. It has been a period similar to that in petrography. The generalizations, the able deductions, have lagged behind. Evidently several geologists have become weary of this "infinite detail" and a mistrust of the microscope seems to have arisen. Although this feeling of being swamped in microscopic detail is a painful experience, nevertheless only through the detail in combination with the large features, and only by utilizing all the related sciences shall we

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[1] It is interesting to observe that in the vanadium-copper deposits in sedimentary rocks, rare elements like vanadium, molybdenum, nickel, selenium, chromium, and silver have been concentrated to a notable degree.

be able to attain the truth. Disaster lurks in the return to the physical methods of pre-microscope and pre-chemistry days. What is needed, but what is not yet in sight, is the master who can combine in his mind the large and the small, the structural features and the story of the microscope.

The following pages contain a brief discussion of that group of mineral deposits that the writer holds have been formed by processes of concentration originating in the molten magmas.

Review of Theories of Various Geologists

The reason why a large part of the mineral deposits, particularly of those that carry rarer elements, has been concentrated by processes of differentiation in magmas, is because in a complex solution like the magma there is most excellent opportunity, as temperature and pressure varies, to split off elements by fractional crystallization, by the development of salic extracts (source of pegmatite dikes), and by gas fluxing. Combined with mineralizers, such as chlorine, sulfur, fluorine, phosphorus, arsenic and boron, the heavy metals will very readily escape from the melt and ascend into cooler rocks where they may condense and form deposits. This theory, brought forward by the genius of Elie de Beaumont long ago when but little was known of the character of the magmas, was for a time generally accepted, at least in France. One is surprised in reading (1)[2] Burat's *Geologie Appliquée* (1853), how fully the genetic connection of igneous rocks and ore deposits is recognized and how clearly the ore zones are outlined.

Later on, the ground gained was lost by the advancing of an elaborate theory of the circulation of waters of surface origin, so that in the views of Daubrée and DeLaunay there is but little left of de Beaumont's generalizations. Later came Sandberger with the theory of lateral secretion and the best that Stelzer and Posepny could do in opposition to him was to uphold the derivation of the ore-depositing waters from an indefinite deeper zone.

Vogt's rejuvenation of the magmatic theory of ore deposits (2) about 1895 was a great step in advance and he soon won adherents in this country. The teachings of Van Hise with his insistence on universal circulation of surface waters and their supposed universal powers of leaching and deposition gradually faded away. At present the magmatic theory holds full sway; many have contributed to it; no one in our country can be regarded as the sole proprietor of it. While it is not intended to present a full history of the magmatic theory outside of this country, no account would be at all complete that left out the name of

[2] See page 92 for literature cited by numbers in parentheses.

Richard Beck, who soon after the appearance of Vogt's papers in 1893–95 became an enthusiastic advocate of this doctrine.

Van Hise in 1896, A. C. Lane about the same time, Spurr in 1898, (3) Crosby in 1897, (9) and others had recognized the transition between pegmatite dikes and quartz veins of magmatic origin. Spurr had maintained the continuity of the series into the gold-quartz veins.

The conception that waters of hot springs were of magmatic origin took form about 20 years ago, and this theory has been extensively discussed up to the present time. Kemp, in 1901, (10) confessed his adherence to the magmatic theory and his belief that magmatic waters had formed most of the ore-bearing veins. In 1922 he restated his views in a notable presidential address. Many investigators have held that these hot waters are the agents of deposition but that their pure magmatic origin is a matter of considerable doubt. Most of them are probably mixed products of magmatic material, circulating surface waters and material dissolved from the country rock. The valuable metals and the rarer ingredients are usually held to have been derived from the magmas.

It is quite certain that the impulse towards the magmatic theory was given by the papers of J. H. L. Vogt in the nineties, though Posepny, himself an opponent of Vogt, and Stelzner had paved the way by assuming a derivation of the hot waters from the "Barysphere." About 1900 the theory was actively discussed among a group of American geologists. The writer remembers these discussions very vividly. They culminated in 1903 at a meeting of the Geological Society of Washington (11) when W. H. Weed contributed a classification of ore deposits of more or less direct magmatic origin. It will be found that this classification differed but little from that now accepted by many geologists. Most veins were believed to have been formed by magmatic waters admixed with waters of surface origin. In the long discussion following, S. F. Emmons and F. L. Ransome did not fully approve of the proposed classification, while Lindgren supported Weed. Spurr presented a classification in which the direct magmatic origin of gold-quartz veins was advocated but which otherwise showed but little similarity to his present views.

The writer, after having acknowledged the probability of the magmatic theory in 1900, ventured, in 1906, to present a division of veins and replacement deposits (12) based on the physical conditions at the time of formation.[3] Though to great degree these zones corresponded to depth, the division was essentially according to temperature and pressure prevailing. The shallow, the intermediate and the deep zones were distinguished, since (13) renamed the epithermal, mesothermal and hypothermal.

[3] It is a small matter, but Spurr is in error when he dates this paper, or the views expressed in this paper, from 1919: The Ore Magmas, 2 (1923), 434.

In 1907, (14) the writer attempted to clarify his position in a paper in *Economic Geology*, in which explicit adherence to the magmatic origin of a large group of deposits was affirmed. In 1907, (4) Spurr outlined his "magmatic theory" and his "zonal theory" in a paper which brought a sharp rejoinder from Ransome; (17) in 1912 (5) he still further elaborated his conclusions.

W. H. Emmons, in 1908, (19) ranged himself with the geologists who hold to the magmatic theory; in 1924 he wrote an important paper (20) dealing with the "zonal theory" or changes in depth of ore deposits.

In 1918, W. H. Goodchild of London presented an important contribution to the subject in a paper entitled "The Evolution of Ore Deposits from Igneous Magmas."

In 1923 Spurr published a book in two volumes, entitled "The Ore Magmas," which may be accepted as embodying his experience and conclusions in regard to ore deposits. This was followed by an extensive series of papers among which those listed at the close of this paper may be regarded as the more important (6).

A brief summary of Spurr's views follows as the writer shall later have to express his own opinions, which differ in many points from those of the author mentioned. This paper is not to be considered in the nature of a polemic. Spurr's consistent presentation of his views is greatly to be admired. The writer has benefited from the results of his many data on ore deposits in different parts of the world and, indeed, agrees with him in many respects. Nevertheless, he has gone much too far and the writer believes that some of his conclusions will not stand the test of the sober judgment of physical chemistry. Lest silence may be deemed acquiescence, the writer has ventured to present his own views.

There will be occasion to write down opinions differing from those of other men, all of them like Spurr good friends of the writer. These opinions may be wrong. Time will tell. But free discussion is the key to progress; free discussion we must have.

Summary of Spurr's Views

The majority of ore deposits are termed "veindikes" because they are essentially intrusive masses formed by the injection of the "ore magma." The ore magma is produced, not during the normal cooling of the igneous rock, but during very slow and long continued differentiation of magmas at very great depths, and they are forced up by "telluric pressure." Volcanic emanations, therefore, do not represent ore magmas. The ore magma is often highly concentrated and viscous but vein solutions also have the property of intimate penetration and replacement of the rocks traversed, indicating in these cases much thinner solutions. As examples of "veindikes" are given, Californian, Canadian and Victorian gold-quartz veins, the Cobalt veins, the Camp Bird veins, the Bolivian

tin veins, the United Verde deposit in Arizona. In fact, it appears that all ore deposits, whether of replacement type or not, are "veindikes," or injected ore magmas.

The escaping gases from the ore magma into the country rock are held to induce precipitation of the metals. Deposits carrying mainly cinnabar and stibnite are deposited from hot-spring waters. These latter solutions are far different from magmatic solutions which have penetrated the rocks as "veindikes." The hot springs are the aqueous residue from real ore magmas which nowhere reached the surface. Spurr differentiates between the siliceous ore magmas derived from acidic rocks and basic ore magmas which are rich in sulfides and poor in gangue materials.

In part, the ore magmas appear to have been gels and this is not confined to the siliceous ore, but also to the pyrite-carbonate veindikes of which it is said that "they were injected as a pyrite-carbonate magma, intruded in a gelatinous state, in a condition between solution and crystallization."

In the classification of ore deposits the gangue minerals, such as tourmaline or calcite, are of little importance. Speaking of Peñoles, Mexico, it is said that the vein fluid up to the calcite stage was stiff or viscous. Hardening of the gel must have swiftly followed intrusion. The gels contained some water which produced silicification of the country rock. A deep plutonic sequence of vein deposition is discerned, which may extend over a space of 25,000 to 30,000 ft. An interesting example is given from Ojuela, near Peñoles, where the ore zone is 3500 ft. deep passing from the chalcopyrite and arsenopyrite stage in depth to the lead-zinc stage and, still further up, to the silver stage; above this stage the veins are barren with siderite and calcite for 2500 ft.

The upper vein zone, within 3000 ft. of the surface, in volcanics, presents a separate and distinct sequence from high-temperature minerals, such as tin and wolfram, to the complex silver ores. The result of rapid precipitation in this zone is that the ores of the different vein zones are quickly superimposed or "telescoped." If rightly understood, Spurr says the ore magma of the upper zone came from the same source as that of the deep zone but held the metals in solution longer, for the magmas which reached the surface arrived there with a temperature hotter than that which they had in depth. The total metal content was precipitated within the upper 3000 feet.

The existence of an intermediate zone is inferred. This would be synchronous with an upward surge of the rock magmas slowly ascending. The rising temperature will cause an inversion of that normal order of deposition of the different metals that takes place ordinarily by falling temperature. As examples of this are given Aspen, Colorado, and Tiro General, Mexico.

Georgetown, Colorado, veins which show a lining of quartz crystals and a central mass of sphalerite and galena are explained as follows:

The fissures were filled by injection by a highly concentrated galena-blende magma. The first precipitation was of quartz deposited before the cooling had progressed to the stage necessary for the crystallization of blende and galena.

Ore magmas may form banded veins. In many veins gold appears near the walls; Spurr has this explanation: "I suggest the action of differential gaseous tension, the metallic elements having, as I assume, a higher gaseous tension than silica.

Wavy, banded structure in veindikes is said to be usually not due to gradual filling but "indicates a pulsating deposition from a standing solution (ore magma?) which filled the fissure."

Inclusions of country rock in veins were suspended in the thick, viscous ore magma and frequently have been carried far upwards in the veins. As inclusions are common in all classes of veins, it follows that the wavy, banded veins just referred to were also consolidated from a thick viscous solution. The veindikes have made room for themselves by their inherent "telluric pressure."

Contact metamorphism as usually defined is a misleading term. The ores in metamorphosed limestone have not been derived from adjacent or upper parts of the intrusive, but have ascended as ore magmas from the same deep chambers where all the ore magmas originated.

The "zonal theory" of Spurr is based on the supposed vertical succession of metals in ore deposits, usually as follows: 1, tin, tungsten, molybdenum; 2, gold; 3, copper; 4, pyrite and arsenopyrite; 5, zinc; 6, lead; 7, silver. This indicates deposition during gradually lowering temperature. A reversal of sequence is obtained by deposition during a stage of rising temperature due to slow upward migration of heat. This closes the presentation of Spurr's views.

Fundamentally, these do not differ so greatly from those of other "magmatic" geologists. I quote, for instance, with hearty approval, this sentence in his first paper[4] in *Economic Geology:*

This theory proposes that metalliferous fluids, from which most ore-deposits are precipitated, are extreme differentiation phases of rock magmas; that most ore deposits and "mineral veins," as a class, represent one or the other of the extreme products of magmatic differentiation; and that the most striking chemical difference between ore deposits is due (in the more important class representing the siliceous extreme), to successive precipitation in theoretically vertical zones, as the fluid migrates toward the surface, and with diminishing heat achieves more and more mature crystallization.

The Magma

Magma has a definite meaning. The derivation is from a Greek word signifying a soft or doughy substance. It is used to designate the material from which all our igneous rocks have consolidated. A simple

[4] J. E. Spurr. *Econ. Geol.* (1907), 795.

melt of an igneous rock would not ordinarily be a magma for the volatile constituents, such as water, sulfur, chlorine, fluorine, and certain amounts of metals entrained made their escape when this igneous rock congealed. It is assumed that large amounts of magma still exist in actual or potential fusion in the crust of the earth.

All the magmas we know are silicate melts with more or less volatile constituents. There may be other worlds where the magmas, the lava flows and the intrusions consist chiefly of iron, nickel, gold, platinum or sulfides. With some stretch of the imagination we may even picture a world in which the magma consists of colloidal gel. But for our earthly purposes we can only define *magma* as a liquid of high temperature that consists of a mutual solution of complex silicates, at times with admixed oxides, and always containing a certain amount of dissolved volatile material. Logical usage demands that the term "magma" be restricted to the above definition.

Isolated parts of the magma such as the gases, or the metallic oxides, or the silica cannot be called a magma and not even an "ore magma." When differentiation has proceeded to a point where only one component or several similar components remain, the term magma should be dropped. One may properly speak of quartz or iron ores, sulfides or gases *of magmatic origin*. It should be obvious that you cannot take a complex whole, cut it up in its integral parts and still apply the same name to the parts as to the whole. The pegmatites are derived from a complex silicate melt and a volatile component and would, therefore, still logically be considered as congealed from a magma. A basic rock with more or less sulfides, such as that from Maine, described by Bastin, may also be said to have solidified from a magma.

It is very difficult to ascertain the composition of Spurr's ore magma. From a close analysis of Spurr's writings it would seem that it is a viscous melt containing useful metals, but frequently it takes the form of a gel—a silica gel or a carbonate gel (if such a thing has permanent existence in nature). On the other hand, evidently a liquid must be extraordinarily tenuous if it can penetrate and replace all rocks without phenomena of filling. About replacement deposits the following is asserted: "Many veins have been formed by replacement or impregnation along fractures, indicating a thinner and more aqueous ore magma." It is evident, therefore, that any vein-forming solution is an ore magma.

Although the writer freely recognizes that a magma may split up in many unlike parts, some of them containing useful metals, he feels obliged to reject the term "*ore magma*" for the latter products.

The Differentiation of the Ore Solutions

What causes the volatile constituents and the silica to separate from the parent magma? Apparently the most general reason given is the

crystallization induced by cooling. P. Niggli has given the best treatment of this process. The salic extracts are first separated with initial increase of pressure. Later the hydrothermal solutions escape with gradual lowering of pressure (22). Obviously, however, a similar escape of volatile substances must result if a magma without crystallizing is gradually forced up to higher levels. Finally, as the magma rises to the surface, emanations are separated from it in the form of complex volatile substances but which rarely are sufficiently concentrated in proper channels to form ore deposits.

It follows then that the differentiation of the volatile part is a process beginning at great depths and not ended till the final congealing at the surface. In abyssal depth where the magma has remained long in equilibrium, no crystallization takes place and no emanations are given off. The equilibrium must be disturbed before separation can take place. Most, though by no means all, of this differentiation the writer holds takes place near the apexes of stocks and batholiths.

Dikes

During igneous intrusions and flowage, tabular masses of magma are frequently injected in the surrounding rocks. These we call dikes. Conceivably the rocks may be sundered by tension and the magma would fill the open tension crack. Usually however, the space is opened by the hydrostatic pressure of the magma aided by the tension of the gases contained. The magma forces the dike walls apart. The magma fills the space created. While fluid it may surge upward to the end of the dike; it may even overflow; but when equilibrium is attained, the freezing begins rapidly and the dike is soon a massive, homogeneous rock. Where the walls are cool, the chilled border may be finer grained or otherwise slightly different. There is no banding; substantially the rock is homogeneous. The volatile substances have escaped upward or sideways through the walls; if present in abundance, contact metamorphism may be evident close to the walls. Lime silicates and iron silicates and ore may form in limestones to a limited extent. The writer is sure that such dikes are not uncommon.

Pegmatite Dikes

Most of us probably agree that the pegmatites with their coarse texture and often complex composition have congealed from a magma that had already undergone a considerable amount of differentiation. These products may be designated as salic extracts in which the silica, alumina and alkalies have been greatly concentrated. They are often called the residual juices of granitic rocks, separating or being squeezed out as the crystallization begins throughout the rock. The silica entrains many of

the common and rare metals such as tin, tungsten and molybdenum, and the rare earths are carried along probably by mineralizing agents for phosphorus, boron, fluorine and chlorine are present in many of the dikes. However, the sulfides are poorly represented, except in some varieties which carry bornite and chalcocite; gold is rare or present in traces only.

It is assumed by some that the pegmatite magma contained much water; others believe it contained but little. At any rate the magma was surely very fluid; the melting points of the silica and silicates had been greatly lowered; the gases contained—or liquids above the critical point—frequently penetrate the wall rocks causing deposition of high-temperature minerals, such as tourmaline or other products of contact metamorphism. The rare minerals (such as tourmaline and beryl) formed at a somewhat later and cooler stage; extensive replacements occurred, as of albite after orthoclase (23) and as still later products we find zeolites, fluorcarbonates and carbonates.

The life story of a pegmatite dike is sometimes complex but it is certain that the great majority of such dikes are compact, simple and contain few rare minerals. Such druse minerals and replacements may have been formed rather by residual liquids still contained in the dike than by new supplies of magmatic juice of different composition from that of the original dike material. Schaller, and others as well, think they result from replacement by later ascending hydrothermal solutions.

Like Spurr, the writer feels sure that the pressure of the magma opened the dike fissures. Evidence of its comparatively great mobility is afforded by the intimate pegmatization of many rocks.

According to Spurr, the pegmatites form direct transitions into gold-bearing quartz veins. Therefore, he says, they came from the very deep magma basins where differentiation has been going on for eons in slowly cooling magmas. The writer would like to present the slightly different version that pegmatites are a normal accompaniment of the consolidation of acidic rocks whether abyssal or intrusive near the surface. We find them, for instance, abundantly in the deep Archean of Canada and at the apexes of batholiths reaching within a few thousand feet of the surface. Therefore, they are a phenomenon of normal cooling not of abyssal differentiation.

Another feature indicating fluidity of the pegmatite magma is seen in the not uncommon banding. This banding is coarse, never fine. Occasionally, one may observe sheaves of muscovite along the walls, then large orthoclase crystals and lastly a central filling of pure quartz sometimes projecting in drusy cavities.

The temperature of the beginning of the separation of the druse minerals (such as tourmaline) may be confidently put at about 575° C.; the earlier temperatures may have been much higher; the final tempera-

tures were 200° or 300° lower than 575°, which represents the inversion point of quartz. The pressure differed greatly; in the Kristiania region, according to Brögger and Goldschmidt, it probably did not exceed 400 at. In abyssal pegmatites it was certainly much higher.

Relation of Pegmatite Dikes and Veins

It is recognized, and fully in agreement with Spurr in this respect, that pegmatite dikes may form transitions into barren quartz dikes and that many gold and tin veins show a close relationship to the pegmatites, but the writer is inclined to think that actual transitions are very rare. At least he has seen none that could be properly so termed. Some of these veins are evidently directly derived from intrusives by differentiation and so are the corresponding pegmatites, but the differentiations seem to belong to two separate, successive stages. In few places is this better illustrated than in Reuning's (24) description of the Natas mine in German southwest Africa. Here is the apex of a small batholith intruded in schists. From the apex radiate many typical pegmatite dikes some of them with a little gold, scheelite and copper ores. There are also later and much longer quartz veins of what is called hydrothermal type, which also carry the same two minerals. No doubt these veins are true exudation products of the magma. Clearly we have here two stages; 1, the pegmatite stage, and, 2, the hydrothermal stage, the latter containing much less alumina and more water than the first.

Spurr speaks of the close relationship of the California gold-quartz veins with pegmatites as shown by the common presence of albite, but he must be aware that the albite here simply means that the country rock was rich in sodium and that this albite cannot be regarded as of magmatic origin. The albite develops by preference in the altered country rock at the Utica mine, Calaveras County, Calif., at the Eagle Creek mines, Alaska, and at Kalgoorlie, and in all places it can be shown that it only appears in force when the country rock is unusually rich in sodium.

The case is similar to the adularia which often develops in epithermal veins in country rock rich in potassium, as at Cripple Creek, at Tonopah, at Silver City, Idaho, and many other places. It does not mean in the least that this vein material is a pegmatite. It is not a magmatic phenomenon at all. It simply forms instead of sericite under certain conditions, the reasons for which we do not yet quite understand. Spurr's conceptions of the California gold-quartz veins are somewhat curious. He says, for instance,[5] that the quartz does not penetrate or silicify the country rock; therefore, it must have been deposited by dry ore magmas; in the country rock the alteration is to iron carbonate. It is general knowledge that the alteration of the country rock is intense

[5] J. E. Spurr: Ore Magmas (1923), 523.

and in places, as along the Mother Lode, very extensive. Also that siderite is not one of the alteration products, only mixed ankerite carbonates, or calcite, with a great deal of sericite, which indicates strong hydration of the country rock from the vein-forming solutions.

The Fissures

The question of how the vein fissures were formed and filled has been debated for at least a hundred years and we are apparently yet far from a unanimous opinion. Fissures may be formed by compressive or by tensional stresses. If the former, the conditions for open spaces to be filled are more unfavorable. On the other hand, tensional stresses will tend to preserve open spaces along the fissure. The writer believes now that fissures frequently result from tension. Even under these conditions the fissure will tend to close under the influence of rock pressure, though less rapidly.

Actually we find that a large number of deposits have originated entirely by replacement, evidently along tight fracture zones, only penetrable by very mobile solutions. In other places we find the strongest kind of evidence of filling of open spaces. It seems that open spaces may well exist to a depth of many thousand feet. Large open cavities are often found in veins. Old mine workings and stopes may often remain open for a long time. Any mining engineer has seen enough examples of this. Take the case of the alpine mineral fissures (25). Although they were formed at great depth, and often carry high-temperature minerals, these flat veins are still open in part perhaps for 10 to 20 m., and for much greater distance before being filled. And the case for the filling is, evidently, perfect in this class of veins. In abyssal depths, fissures will not remain open unless strongly supported. In Scandinavia, for instance, where the Archean was deeply buried, there are very few examples of filled veins.

The Filling

When the fissures were open they were filled with solutions. These solutions may have been under a very heavy hydrostatic pressure if they communicated with the surface, say 30 at. per 1000 ft. If the solutions emanated from a magmatic source but did not connect with the surface the pressure may have been several hundred atmospheres, not enough perhaps to break through the crust but, sufficient to counteract materially the closing tendency of the walls. The net pressure would create near-surface conditions, and everyone knows that in epithermal veins abundant open cavities did exist.

Even when openings did connect with the surface they might have been so limited as to produce a throttling of the magmatic gases or liquids and long continued heavy pressure.

The writer therefore fully agrees with Spurr's view that the fissures were held open, in some deposits, by internal pressure which continued in the main until the fissure was filled; but disagrees with him as to the mode of filling.

Spurr's typical "ore magma" for the gold-quartz veins is apparently a gel containing about 14 per cent. SiO_2 and 86 per cent. H_2O. Such gels have been made but they are very unstable. Ordinary gels contain more nearly 3 per cent. or 4 per cent. SiO_2. When we further consider the hydration of the country rock by the development of sericite, this would still further increase the water in the ore magma, and the usually abundant CO_2 as indicated by carbonatization would still further augment the fluidity. So we see that on Spurr's own data the "ore magma" would be pretty tenuous, though probably gelatinous enough to interfere with free crystallization. We would expect to find, as the result of crystallization, a maze of doubly terminated quartz crystals; this is just what we do not find but rather crystals attached to walls and inclusions. The crystallization would leave the "veindike" as a mass of loose crystals; it is, therefore, necessary to have new supplies of "ore magma" constantly ascending in order to create a filled vein. His conception, therefore, in reality differs but little from the old views of gradual deposition by mobile solutions.

Other "veindikes," like those of Georgetown, Colo., are described as being filled by the injection of a highly concentrated galena-blende magma. Just what is meant by this is difficult to interpret. From other passages it may mean a mixed gel of silica and sulfides. From this mixture the silica *diffused* (!) to the walls, there to crystallize in comb-form while the colloidal galena-blende filled the center. Even if concentrated, this sulfide gel could only have contained a few per cent. of sulfides, and how to get the solid filling without a continuous supply of more silica-sulfide gel continually welling up from the depths remains a mystery.

The banded epithermal veins are indeed a great cause of annoyance to Spurr, but when he suggests that this banding is not due to gradual filling but to a rhythmic, pulsating deposition from a standing solution which filled the fissure, then the critic is reduced helplessly to asking how it is possible the fissure will be *filled* from this standing ore magma. One may conceive a gel crystallizing with strong contraction to chalcedony or to more coarsely crystalline quartz; likewise one may conceive diffusion banding created in the gel by reactions between electrolytes. It is needless to add that this banding is not the kind characteristic of banded epithermal veins.

The Camp Bird vein is a case in point. The excellent plates of Ransome (18) so convincingly show the gradual, in part colloidal, deposition from moving solutions that no further argument is necessary.

The writer feels compelled to place in the same categories the statement about the "pyrite-carbonate gels" (Ore Magmas, p. 141) and the "earthy gangue veins" (p. 794) which have not been deposited by water but have been intruded as jellies or gels. No physical chemist has yet been able to produce a stable carbonate gel. In the face of great difficulties the Cobalt, Ontario, veins are also unhesitatingly declared to be of intrusive origin.

However, ordinary *filling*, whether by Spurr's or any other theory, is not the only way in which the "filled veins" have been accounted for. A fairly large contingent, represented by Stillwell (26) and Howe, (27) working respectively in Victoria, Australia and California, assert that what Spurr and many others, among the writer, call by that name is simply replacement in quartz, replacing country rock. The field relations alone speak loudly against such a view; it is not necessary to enlarge upon this; the inspection of photographs representing the veins of Bendigo, Ballarat and California, suffice to make this plain. Frankly, the writer cannot accept a theory of replacement for these veins as far as the quartz is concerned.

Replacement there is, and plenty of it, but it is of a different kind, involving carbonatization, sericitization (with hydration) and, in places, albitization. This proposition is advanced with considerable confidence: That aluminous rocks (shales, schists and igneous rocks) cannot be replaced by coarse-grained quartz without leaving abundant traces by structure, texture and relics. A quartz replacement of an aluminous rock can always, I believe, evidently, be easily identified as such. Until this proposition is proved wrong, the writer will not further enlarge on the subject. A few years ago he ventured to publish a note (15), which seemed to present conclusive evidence of quartz filling based on a specimen from Bendigo.

There is also another clan which places strong faith in the linear force of growing crystals and the ability of this growth to push vein walls apart and thus make room for the vein filling. Recently, Taber (29) has been the most prominent champion of this view, but there are many others apparently overlooked by Taber. The originator of this theory seems to have been von Weissenbach, about 1836. Bischof, Volger and others had similar views. In fact, the latter investigator (30) experimented with copper sulfate and showed how fibrous veins of this material were able to break earthen vessels. Recently, Boydell (31) has discussed this problem from several viewpoints. The writer fully agrees with his views and believes that this force is a negligible factor as far as the opening of veins under heavy pressure is concerned, but that it is probably of some importance in effecting changes and movements in the opened fissure.

Some years ago W. Bornhardt (32) described in great detail the remarkable probably mesothermal vein system of Siegen, Rhenish Prussia, and presented a wonderful series of colored photographs of the veins. He came to the conclusion that the fissures were opened by "internal pressure," but left undecided whether this was caused by crystal growth or by pressure of the solutions. In an appendix, P. Krusch states that he can find no evidence of pressure in the structure of the crystals except very locally, in the veins. Both men appear to lean towards the latter alternative.

"Ore Magmas" and "Veindikes"

In a previous paragraph the objections to the term "ore magma" were stated. According to Spurr the product of solidification of the "ore magma" is a "veindike." All ore deposits connected with magmatic emanations (except cinnabar and stibnite deposits) are evidently "veindikes." A careful study of Spurr's writings can lead to no other conclusion. The "ore magma" is differentiated from the parent magma, injected, consolidated. Nothing could be more simple.

Replacement, or metasomatism, is an awkward fact to face in this theory, and Spurr touches lightly on it. But this much seems clear that he regards pure replacement deposits like that of the United Verde mine, in Arizona, as "veindikes."

Some magmas or fluids of magmatic origin may fill fissures close to the original source without appreciable metasomatism. If it is desired to call these deposits veindikes, well and good. But in practically all ore deposits replacement of the wall rock play a part as important, or more so, than the process of filling. Take the case of the United Verde; this gigantic ore pipe with diameter of 500 ft. is entirely a product of replacement.

From the moment that the magmatic emanations attack the country rock an exchange of constituents begins. The "ore magma" is an ore magma no longer; it is a mixed product of emanations and material such as silica, sodium, potassium, masnesium and calcium, abstracted from the rock, be this part of the batholith or adjoining rock. Usually a little more is added than abstracted, say on the order of 200 to 300 kg. per cubic meter. For the constituents lost, carbon dioxide, sulfur, iron, copper and other substances have been added. It is not difficult to realize that after the solutions have ascended the fissure a short distance the so-called ore magma loses its magmatic nature in great part. Besides water is undoubtedly acquired especially when the solutions ascend in the rocks surrounding the intrusive. The writer has a strong conviction that much of the barren material deposited in the upper parts of the veins is simply derived from the country rock lower down. Of course, when the

whole deposit is of the replacement type, the change in the nature of the solutions is correspondingly rapid.

Let us see what happened in case of some Bolivian tin veins that Spurr includes among his "veindikes." At Caracoles (16) the veins are of the hypothermal[6] type and are formed entirely by replacement. There is no filling to speak of. The solutions must have been very mobile to penetrate the tight fissures and adjoining rock; they must have ascended from the lower part of the granite batholith into the upper cooler and congealed portion. They contained much water, as chlorite and sericite are among the chief metasomatic products. They were loaded with iron, tin, boron, magnesia and some sulfur. They abstracted sodium, potassium and calcium to the extent of 200 kg. per cubic meter from the slightly altered country rock. From the vein itself much larger amounts per cubic meter must have been extracted. To call such solutions ore magmas is surely to convey an erroneous impression.

The veins at Chocaya afford another example. Here the tin veins are of the epithermal type and have formed in acidic flow rocks. Here we find pyritic and sericitic alteration of the country rock, but there is no ore in it. The vein is a beautiful example of a filled cavity; in places full of touching fragments of country rock (a bubble) which are coated with pyrite, quartz and cassiterite in thin colloidal layers repeated three times. Later follows chalcopyrite (oldest), stannite, tetrahedrite, chalcopyrite and jamesonite as successive wavy bands filling the fissure. The succession is the same as at Potosi. The open fissure has been gradually filled.

All this will make clear my reasons for rejecting the terms "ore magmas" and "veindikes," except as they may refer to certain pegmatite dikes and some very closely allied forms of quartz veins.

THE INCLUSIONS

Since 1836 when von Weissenbach described and figured the Saxon deposits, the inclusions found in fissure veins have attracted the attention of observers. Many of the older writers of textbooks (Stelzner, Posepny, Beck) call attention to the loose, partly filled rubble of inclusions of country rock that locally fill many veins and describe the partical coating of these fragments by gangue and ore. Frequently these inclusions appear to be swimming, unsupported, in the vein filling. Attempted explanations of this phenomenon are numerous. The first suggestions would be that the local vein section is deceptive; that the fragments seem unsupported but really touch at points beyond the section. Experiments have been made to test this by pouring fluid cement into a box

[6] The word hypothermal of course involves no commitment as to the source of the hot water. Most of it probably came from the magma but the word itself implies nothing but an aqueous solution of high temperature.

of such loose rubble, and preparing sections after hardening. Such sections actually show many unsupported fragments (33).

Still others maintain that the inclusions have been separated from the walls by the "force of growing crystals;" others again that they are remnants of country rock which have escaped the general replacement in the vein. Spurr believes that the inclusions are supported by the viscous "ore magma," and thinks that they may have been brought up by this "magma" from indefinite depths. He even goes so far as to state that the presence of unsupported inclusions is a proof that the vein was congealed from the "magma."

Any of these explanations may be applicable. Surely the crystallizing gangue often detaches fragments from the walls; such results are often apparent in the Cobalt veins. In these veins which are closely related to the pegmatites the solutions may be able to support fragments. No one may doubt that residuals after replacement may closely simulate inclusions (34). But, as the writer believes that the crystallization of the vein matter proceeds generally from solutions of no great viscosity and is a gradual affair in the rising fluids, it seems clear that an important factor is the constant dropping off of fragments from the walls. These fragments will be caught and supported by the vein material in process of formation.[7] Frequently the writer has observed inclusions of wall rocks coated by a loose comb of quartz crystals or incrusted by a shell of rich ore minerals. This recalls the "cockade ore" in which the inclusion is partly replaced and surrounded by rings of successively deposited sulfides. Fine examples of this occur in the Alice mine, Butte, where these inclusions in part are made up of older pyrite-sphalerite and where they are contained in a drusy matrix of rhodonite, rhodochrosite and quartz.

That the vein solutions have carried up fragments from great depths does not appear demonstrated. Howe(28) states that in the Grass Valley veins, the intersection of contacts by the veins are marked by a change in the character of the inclusions. The case from the Camp Bird vein mentioned by Purington and cited by Spurr (7) is not convincing for it is evident that the flow rocks, which here constitute the walls, may have carried up fragments from deeply buried sediments and that such fragments may have been detached during the vein formation.

The Ore Zones

Vertical Succession

The question of ore zones, or the vertical succession of certain metals and minerals in ore deposits, is of great interest and importance. Many of the fundamental facts have long been known. Most familiar is the

[7] This explanation, long in mind, was recently advocated by J. W. Young in a letter to the Engineering & Mining-Journal Press, Oct. 13, 1923.

example of Cornwall tin veins, which carry copper ores in their upper parts, in the slates, presumably cooler ground. That lead ores do not always continue in depth but are replaced by pyrite, sphalerite and siderite is also well known. Other examples, at Bingham for instance, show veins that in the unaltered limestone carry mainly lead and zinc but go over into copper ores within a vertical distance of 1000 ft., while the quartz-calcite gangue changes more or less to lime silicates.

A case has been described by Spurr[8] from Penoles, Mexico, where the veins present a continuous deposition from the high-temperature stage to the stage of the rich silver ores, and finally to the last deposits of nearly barren gangue minerals. Many gold-quartz veins continue without much change for 4000 or even 6000 ft. of vertical depth. Evidently the problem is complicated and one in which exceptions and reversals are plentiful. Generally speaking, the high-temperature veins are likely to carry tin, tungsten, copper and zinc. But on the other hand, the Broken Hill lode (N. S. W.) is one of the world's greatest lead deposits though undoubtedly formed at high temperature.

Gradual lowering (or increase) of temperature is the universal explanation offered by Spurr who gives the following theoretical vertical succession brought about by lowering of temperature. Deepest: *A*, tin, tungsten, molybdenum; *B*, gold; *C*, copper; *D*, pyrite, arsenopyrite (auriferous and argentiferous); *E*, zinc; *F*, lead; *G*, silver. The first two divisions are said to be connected with coarse-grained rocks; the second and third with fine-grained holocrystalline rocks; the last three with fine-grained porphyritic rocks.[9] This latter coordination scarcely seems intelligible. A gradually increasing temperature brings about a reversion in the order; thus at Aspen, polybasite is said to be followed by galena and that by zinc blende. In the epithermal (near surface) veins the deposition is stated to proceed under rising temperatures and, according to Spurr, the whole succession comes down quickly, superimposed and without order.

Undoubtedly this idea of "telescoping deposition" and reversal of order is a good one, but it would seem to be far more easily applicable to a gradual deposition by solutions than to a thick, viscous congealing "ore magma."

The order of deposition is not simply a function of temperature. The precipitation is a function of temperature, pressure, time, concentration and accompanying components in the system. Therefore any *dictum ex cathedra* regarding the succession in vertical zones will be difficult or even impossible. Cassiterite is generally a high-temperature mineral, but it may be abundantly deposited in mesothermal (Potosi, Bolivia),

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[8] J. E. Spurr: The Ore Magmas (1923) 53.
[9] J. E. Spurr: *Op. cit.*, 18.

or epithermal veins (Chocaya, Bolivia); molybdenite is normally a high-temperature mineral, but it is present in finely divided form in the Cripple Creek veins, which are also epithermal. This does not indicate any reversal, simply that the initial temperatures were higher than those of the main deposition.

The data of vertical succession are in part obscure and contradictory; the main known features have been set forth by Spurr and also, lately by W. H. Emmons (21), who has devoted an important paper to this subject. The great vertical range of the mesothermal and hypothermal gold-quartz veins stands out prominently, contrasted with the short range of the epithermal gold-silver veins. Other things are more doubtful. Why are copper ores followed upwards by zinc and lead, whereas usually zinc separates out before rather than after copper?

Succession in the Same Ore

Another closely related and interesting subject is the succession of minerals in the same ore. In all deposits of magmatic affiliation, from contact metamorphic ores to those of the epithermal veins, the same general succession holds, and the later minerals invariably replace the earlier ones. Not all of the minerals, of course, are present in the same ore, but broadly speaking this succession holds (beginning with the older): silicates, quartz, carbonates, and other gangue minerals (sometimes overlapping and repeated), magnetite, specularite, pyrite, arsenopyrite (with Ni and Co arsenides), wolframite, cassiterite, molybdenite, bismuth, bismuthinite, pyrrhotite, pentlandite, stannite, sphalerite, enargite, tennantite, tetrahedrite, chalcopyrite, bornite, galena, chalcocite, carbonates, argentite, ruby silver, polybasite, chalcopyrite, electrum, lead sulfantimonides, carbonates, calcite. In general sphalerite antedates the copper minerals.

The position of gold is not constant; usually it is late in the series and may be accompanied by tellurides. The succession is in places reversed or changed. Magnetite may follow pyrite and sphalerite may follow galena; sometimes pyrite and marcasite are repeated. Spurr believes this indicates rising temperature; it may be so, but this simple explanation is probably not the only solution of the problem. The development of almost any kind of ore then involves a complex but orderly and far-reaching series of replacements. The writer submits the belief that a thick and viscous "ore magma" is incapable of effecting these replacements.

Temperature of Deposition

Spurr advances an explanation for the great richness of the bonanza ores in the epithermal veins; they are derived from the same abyssal

magma zone as the rest of the "ore magma," but they are deposited under conditions of increasing temperature as the lavas and the "ore magmas" rise to the surface. Aside from many other objections to this, the succession of minerals in the ores from these bonanza zones is substantially the same as for all other ores, and they would, therefore, seem to be formed under decreasing temperature.

Compelled by overwhelming evidence, Spurr admits that cinnabar and stibnite may form from hot spring waters, close to the surface. But at many places (National, Nevada and other districts) these veins pass into gold and silver veins. Moreover, neither mineral is confined to the epithermal veins.

Recent statements by Spurr (38) indicate that he believes that all ore deposits of magmatic affiliations have been deposited at minimum temperatures of 365° C. and ordinarily about 400° to 500° C. No elaborate attempt will be made here to refute this view; but if this were correct, all deposits in limestone would have a gangue of lime-iron silicates. Ore deposits, the writer maintains, develop during subsiding temperature and may well be formed at temperatures of 100° to 150° C.

The next great task for the geochemists is to explain this remarkable general succession, which as stated, broadly holds for all deposits ranging from pyrometasomatic to epithermal. A. C. Spencer has attempted to explain it for the deposits at Ely; great credit to him for this, though his solution cannot be considered final. It would seem that temperature is the most important factor; certainly it is not the only one.

Contact Metamorphism (Pyrometasomatism)

The manifestations of contact metamorphism, meaning thereby the replacements and crystallizations induced in sediments or similar susceptible rocks at intrusive contacts, are so well known and have been described so fully that it is not necessary to enter in details. All observers agree that the alteration of clay shales to hornfels and of limestones to silicate rocks are the result of igneous emanations at first more aqueous, later more diversified. At the first contact the processes begin and reach their peak when the intrusive solidifies. The strongest action was produced when the rock at the contact had solidified and the interior was in the process of congealing. The common distribution of tourmaline, scapolite and pyrite, sometimes far from the contact, indicates how widely the adjacent rocks were permeated by the hot fluids, for these minerals contain mineralizers like boron, chlorine and sulfur.

The introduction of iron and silica in garnet, wollastonite, epidote and hedenbergite is so common that it scarcely needs further emphasis. It is also extremely common to find copper, zinc and lead minerals scattered in the lime-iron silicates and slightly later than these, often simply scattered, often in larger masses of economic importance.

Spurr doubts the existence of "contact metamorphic" deposits. To his mind such ores are simply deposited by "ore magmas" rising on or near the contact from the deep manufactories of such "magmas." Every little contact metamorphic deposit is connected with the depths.

Undoubtedly veins exist that have been formed in limestone by deep-seated emanations. The writer has seen many of them; Spurr has described fine instances from Mexico. The only condition needed is that the temperature shall have exceeded say 500° C., below which limit the lime silicates can no longer form.

But the writer has also seen so many instances where no such deep-seated fissures can be postulated, that it seems absurd to deny the actuality of emanations from adjacent intrusions. The localizing of such deposits at limestone inclusions and projecting points can hardly be explained always by convenient deep vents of "ore magmas." The writer cannot understand how anyone who has read Harry Eckermann's (35) important work on the contact metamorphism of inclusions can explain these facts in any other way but by emanations from crystallizing magma. Sometimes, as at White Knob, Idaho, the alteration has gone so far that limestone and igneous rock can no longer be sharply differentiated. Why call on abyssal sources when the conditions at the top, or apex, of any batholith are well known to be most favorable for the collection of liquid and gaseous emanations.

The admirable map of Copper Mountain by C. W. Wright (36), offers the writer convincing proof of his views. Here the upper part of a batholith is several miles in diameter. At every place where limestone is at the contact, there is contact metamorphism, and at close intervals all along these parts of the contact, the copper deposits are located within the narrow strip of silicate rocks. Did Nature provide a fissure or vent at each one of these deposits, connected with the mysterious depths whence the "ore magmas" came?

The gases, evidently, came from the congealing intrusive; not only from the immediate contact but from large volumes of magma adjoining it, laterally and in depth. In origin one can not separate the calcium silicates, the iron silicates and the sulfides and oxides. Though appearing in orderly succession they all came from the same source.

Conclusions

This paper begins with a brief historical review of the magmatic theory which states that many ore deposits have been formed by magmatic emanations. A special review is given of the opinions of J. E. Spurr and his use of the terms "ore magmas" and "veindikes."

The term "ore magma" is examined and rejected except for special cases, for instance, a normal magma containing much sulfides and oxides,

or pegmatite dikes rich in the same constituents. After a brief consideration of dikes in general, the pegmatite dikes are discussed; under circumstances they may carry ore minerals and sulfides, also a little gold, but there is no unbroken succession between pegmatites and gold-quartz veins or tin veins. The dikes were forced open by the pressure of the magma. The presence of albite and adularia in veins does not indicate a relationship to pegmatites.

The fissures of the hydrothermal series of emanations often contained open spaces, especially where broken open by tensional stresses. The hydrostatic pressure from the surface or the pressure of escaping emanations, or both, helped to hold them open. The linear force of growing crystals is not considered an important factor in vein formation, except for the detaching of fragments in the opened fissure.

Fillings were deposited gradually by mobile solutions, not from thick and viscous fluids, though close to the magma the viscosity no doubt was higher than nearer to the surface.

It is asserted that normal, coarse-grained vein quartz cannot be formed by replacement of aluminous rocks. The great importance of replacement in vein formation is emphasized. It is stated that the emanations begin to change with their first contact with cooler rocks; that they are continually depositing their magmatic load and taking up large amounts of material from the country rock.

The term "veindike" is rejected as misleading, except for certain pegmatite dikes and other exudation products of purely magmatic origin.

The view that inclusions of country rock in vein matter are carried up and supported by a viscous vein-forming "ore magma" seems improbable. The effect of the force of crystallization detaches them from the walls or they fall into the fissure from shattered walls and are supported by the partly filled vein.

The so-called zonal theory is examined after a brief historical review. It is pointed out that the vertical succession although true in a general way, presents many inconsistencies and reversals, which cannot always be explained on the supposition of an increasing temperature. Closely connected with this subject is the succession of minerals and the succession of replacements in the same ore. An attempt is made to give a general succession of this kind. It is shown that the problem really is very complex and that it can only be solved by the physical chemist. At any rate a thick and viscous "ore magma" is not capable of producing this striking succession of replacements.

Contact metamorphism (pyrometasomatic ores) is asserted to be a universal phenomenon attending the solidification of magmas. Both lime-iron silicates and sulfides have this origin. It is not denied that emanations ascending in limestone from deep sources may produce similar effects. Copper Mountain, near Ketchikan, Alaska, substanti-

ates the views given; also the metamorphism of limestone masses included in the igneous rock.

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DISCUSSION

J. T. Singewald, Jr., Baltimore, Md.—I think one of the most suggestive points of Mr. Spurr's is the explanation that the theory apparently gives for the included fragments of country rock in veins, which economic geologists have never very satisfactorily explained. If we reject Mr. Spurr's theory, that problem is still before us, and I would be glad to hear what Professor Lindgren would offer as an explanation in accordance with his views.

Waldemar Lindgren.—Such inclusions have many different origins. Apparent inclusions can be caused by replacement. A rubble of fragments may fill the fissure; that often happens. Different geologists have tried to account for these inclusions by making them experimentally. Professor Graton at the meeting of Economic Geologists last year showed sections of solidified pieces of a mixture of loose rock into which liquid cement had been poured; the inclusions no longer touched.

Barnhardt of the Prussian Geological Survey in 1912 made that same experiment. What we see in a vein is partly deceptive. Any section may show that the inclusions do not touch whereas in reality they do touch.

Another way to account for these inclusions is that the vein during formation is partly filled by newly deposited gangue minerals. The least movement shakes down country rock. A mass of country-rock fragments is continually dropping down in this partly filled vein; this phenomenon, I believe, is often responsible for inclusions.

I do not believe that inclusions have been carried up by viscous fluids or magmas, except perhaps in pegmatite dikes, which are very close to the condition of magmas. Mr. Howe, of whom I spoke before, examined just such a critical contact in Grass Valley, grandiorite below, diabase above, I think, and where the vein went through the contact, the character of the inclusions changed. So you cannot account for all inclusions by one explanation.

W. H. Emmons, Minneapolis, Minn.—Bearing on that point, I would like to call attention to certain fragments in the gilsonite mines of northwestern Colorado and northeastern Utah, particularly in the Black Dragon mine, where the features which Dr. Lindgren has brought out are as clearly shown as in any place where I have seen them.

There, where a gilsonite deposit forms one of the widest and most persistent veins I have ever seen, along the sides of the vein next to the walls, are sandstone fragments that seem to float in the gilsonite and that are about twice as heavy as the gilsonite and could not be regarded as having been held there by the petroleum before it had become gilsonite. There is no evidence of drag near the fragments in the gilsonite. The

vein was filled with petroleum which solidified to form gilsonite. Later the fissure was reopened by planes which at places passed into the sandstone walls. Subsequently these later openings were healed and the fragments remained in the veins completely surrounded by gilsonite. The same process commonly operates where ore veins are filled.

In many deposits the process of replacement has taken place several times. In the great lodes of Ducktown, Tenn., certain parts of the lodes have been in five different states: The lodes were once limestone; later they were replaced by heavy silicates and pyrite; still later the heavy silicates and pyrite were replaced by pyrrhotite, sphalerite and chalcopyrite; later still this ore near the water level was replaced by chalcocite ore, which finally on oxidation became the limonitic gossan. The last two replacements were brought about by cold solutions that enriched the deposits. It seems to me that some advocates of magmatic veins admit replacement as a process of "magmatation," but certainly these successive replacements can be brought about only by the work of aqueous fluids. The process is totally different from the process of assimilation of rock matter by lava or molten rock.

J. C. Anderson, Tucson, Ariz.—Lindgren says that we must absolutely rule out the possibility of the development of vein by the force of crystallization of the growing crystals. But, and this answers in a measure Singewald's question, everyone of us has seen veins in which interlacing veinlets of mineral-bearing quartz have filled radiating fractures in the vein, going out of the main fissure and coming in again, thus surrounding and gradually engulfing fragments of the country rock. A continued growth of the vein and the veinlets might so completely separate them from the wall that they would appear as fragments falling from the walls and floated in the vein before crystallization. Such fragments would tend to be angular, with sharp edges corresponding to the fractures originally followed by the first veinlets.

If we must completely set aside the possibility of such a growth of quartz, I would like to ask Dr. Lindgren how we can account then for the "eyes" of quartz that we all have seen in schist. I recall many instances where a lens or bunch of quartz, sometimes very small, sometimes of considerable size, has completely forced the schist out of its plane, so that the schist curves right around that "eye," meeting above and below at a barren fracture or tiny veinlet. The only possible force that I can conceive of as having caused that curvature of the schist would be an internal expanding force from the growth of the quartz.

Waldemar Lindgren.—In the example cited two different movements are involved. The first is the movement of inclusions, or other material in the open fissure, by the force of growing crystals. I admit that. In an open fissure where fragments are attached loosely to the

country rock, the force of crystallization may very well be able to force such a piece out into the fissure. What I maintain is that this force alone cannot open the fissure.

The second movement is different. I cannot go into it here, because it involves too much. It is a mooted subject, I admit. But I have not yet seen an example for which some other explanation could not be given. It is like the question as to whether crystals in schist have been formed by replacement or by pushing the schist or slate aside by the force of the crystallization. I am quite sure that usually the crystal was formed by replacement, and that the subsequent rotation or pressure from the shale induced that apparent deceptive appearance of the slate or the shale wrapping around the crystal.

This is not quite identical with the example of the quartz eyes, but it would take too long to go into that now. I have confined remarks to a vein thousands of feet long perhaps and thousands of feet high. Can the growing force of crystals push that aside. I believe not.

J. F. Kemp, New York, N. Y.—In quartz veins productive of gold, with associated sulfides, chalcopyrite, arsenopyrite, pyrite, galena, zinc blende, and perhaps some rarer sulfides, often the quartz vein has obviously been first formed and then shattered. The sulfides and the native gold have come into the little cracks and fissures and now appear with irregular, triangular cross-sections in the specimen. Clearly in their precipitation, they have followed systems of cracks.

The filling of such a vein and its values must then be assigned to two periods, one of solid quartz without much value in it; and afterwards, one of enrichment by the circulation of solutions that brought the metallic minerals into the cracks and deposited them.

These relations will recall those of the copper-bearing pyrrhotite and pyrite bodies in metamorphic rocks that are so often found in lenses and sometimes in veins. Our study of polished surfaces has shown that the entrance of the copper followed the precipitation of the massive iron sulfides. Often one can trace little veinlets of chalcopyrite, or some other copper mineral, in what appear to be fractures in an older deposit of sulfides. When we discuss magmatic fillings, we can hardly overlook these obvious solution phenomena.

Other exceedingly significant phenomena that we ought to consider in connection with vein fillings are contact zones, especially those produced in limestone around an intrusive. As nearly as we can interpret them, an emission of some kind of high-temperature silica must have entered the wall rock and have carried iron compounds with it, giving rise to high-temperature epidote, vesuvianite and garnets, especially iron-lime garnets. The limestone must have been penetrated either by gases or by very tenuous solutions that worked their way through the limestone and changed it to high-temperature silicates.

Most of these silicates have never been made artificially. The inference would therefore be that the temperature and the pressure has been high. The silicates are followed evidently by copious emissions of iron, which give the great bodies of magnetite which have subordinate amounts of specularite sometimes intergrown with them. After these came the veinlets of sulfides, particularly those carrying copper, that penetrate the silicates and the bodies of iron ore.

In some veins, such as the bed-fissure vein in the bottom of the Yampa limestone in Bingham canyon, crushed ground can be followed over a quarter of a mile from the lime silicate zones next to the intrusive, through the full extent of contact metamorphism, right into the unchanged coal-black limestone. With the high-temperature silicates are found almost exclusively pyrite and chalcopyrite, which continue across the zone of contact effects. But as the contact effects die out, zinc blende replaces the chalcopyrite. It also dies out shortly and gives place to galena, the last of the sulfides to form.

I do not see in such a case how we could escape the conclusion that mixed solutions have come out and have dropped the minerals according to the physical conditions that were appropriate to each. This explanation is easier to understand than if we consider the ore-bearing solutions magmas that were forced out from an igneous center and worked their way along the fissure.

It seems to me that for the interpretation of what comes out of the magma and what form it is in, contact zones supply very significant evidence. If instead of limestones next to the igneous rock, there are quartzites or some other such rocks, which do not so readily yield to chemical changes, replacements and reorganizations, then the solutions might migrate far from their source, especially if they come out with great force or penetrating power.

R. J. Colony, New York, N. Y.—I am sorry that Dr. Lindgren has decapitated quartz veins from the pegmatite bodies. Perhaps I misunderstood him. In this vicinity, we have, I suppose an infinite variety of pegmatites and we have excellent opportunity to study them. Moreover, intrusive into the Fordham gneiss are igneous bodies that develop pegmatites within their own mass. These we have been able to follow finding that in many places they actually took on a dikelike character and one could trace the dropping out of feldspar from those small dikes and the gradual transition from feldspar into quartz only. It seems to me, therefore, in that connection at least, the quartz veins or dikes, if you like, are very intimately connected with the pegmatite stock, that is, the body of the material in general.

At Columbia it has been our conviction, judging from the material that we have studied from time to time, that the natural course of differ-

entiation towards an acid extreme results in a concentration of quartz and an acid feldspar, which is generally albite, that even may intrude or encroach upon its parent rock. I have also seen more or less basic feldspar penetrated by veinlets of quartz albite, which seems to me distinctly related to the differentiation phase; that is, such structures and mineral relationship indicate albite enrichment towards the close of differentiation. I dare say that if that became prominent enough one should find dikes and veins that consisted entirely of quartz and albitic feldspar.

A. M. Bateman, New Haven, Conn.—I am particularly pleased that one so competent as Professor Lindgren has taken up the viewpoint of perhaps a great number of geologists who have held that veindikes cannot be made to account for all phases of ore deposition.

Concerning veindikes, I should like to remark that a first conception of them given us was that they do not cause replacement. Then later they were made to account for the replacement that we have normally thought due to solutions.

I have often wondered, that if replacement were caused by dikes or ore magmas that are assumed to be relatively viscous, would not one expect to find some of the materials of the wall rock, or of previous vein matter that had been replaced inescapably caught and entangled in such a viscous mass. I should particularly like to hear Professor Lindgren's viewpoint of that question.

Waldemar Lindgren.—Professor Kemp spoke about a contact metamorphism. I don't think he can have read Spurr's books very carefully or he would know that his contact metamorphism or deposit was formed by the intrusion of an ore magma from some point thousands of feet below the deposit.

As to Professor Colony, I disclaim any intention to deny the existence of quartz veins or quartz dikes in the continuation and in connection with pegmatites. Van Hise, you may be surprised to learn, was the first one who advocated that. Lane, I believe, at the same time or before, Crosby advocated it in 1897, and in 1898 Spurr brought back his results from the Yukon which substantiated these views. I know there is albite in pegmatite veins. I know albite also occurs in many other places. But the albite per se does not prove anything. Albite in a metalliferous vein is of different origin from albite in a pegmatite dike and from albite formed in certain limestones which have been metamorphosed but very little. What I question is whether anybody has ever seen a pegmatite dike running over in continuation into a tin vein—a tin vein, mind you, not a tin dike—or a pegmatite dike running over into an economically important gold quartz vein.

I would like to remark that the zonal theory is very old. You will be surprised how clearly the underlying conception is outlined in Burat's Geology which was published in 1853.

W. H. Emmons.—I wish to say in defense of the zonal arrangement that if we subdivide lode deposits into six groups classified with respect to their positions on the roofs of their parent batholiths, we find that the zonal arrangement can generally be recognized in one group, that it is common in three others and that it is probably everywhere wanting in two others of the six groups. In a certain group of deposits formed at shallow depth, this arrangement is generally lacking or imperfectly shown. Most of these deposits are in areas of closely spaced dikes. They are not known to be associated with batholiths, but there are reasons for supposing that a batholith underlies the area and deeper erosion would probably reveal it.

In a second group called the acrobatholitic, the cupolas, or highest points, of the batholith are exposed. Ore deposits are commonly found in and around these cupolas. They are the most favorable situations for lode ores and for systems of zonally arranged ores. Further erosion of the batholith will reveal larger areas of the partly eroded roof of the batholith, and in this, the third group, zones are also common. At a later stage, represented by the Sierra Nevada in the Mother Lode region, California, zones become less common; and in the fifth stage, where ores are found in roof pendants surrounded by the batholith, no zones are yet recognized. In the sixth stage of erosion, even roof pendants are removed. At this stage ore deposits are rare and zones are wanting.

The well defined zones are confined to the regions around, a little above and not far below the cupolas. To me that indicates a progressive precipitation from certain centers outward and upward. Does it not show that ores are formed by aqueous solutions in the main, at least around the higher points of the batholith? It is possible that at very great depths we may find something approaching the true magma more closely, but certainly the zonal arrangement itself shows, I think, that progressive precipitation from aqueous solutions is responsible for the ores that are formed in zones surrounding the summits of cupolas of batholiths. That, after all, is the preferred position for ore deposits. That is where we find the disseminated ores of copper and tin. That is where we find the succession of zones. That is where we find our most richly mineralized areas, with few exceptions. Gold deposits only are most abundantly developed in roof pendants and in deeply eroded batholiths.

At the last stage, which is still deeper, gold itself is gone and little or no ore remains, because ore deposits are features of the roofs of batholiths and are rarely ever found far inside the batholiths.

The cryptobatholithic deposits, where the bath is hidden, are the most puzzling deposits.

Waldemar Lindgren.—I would like to ask Professor Emmons where these ore solutions came from. Spurr said, I believe, that they were differentiated in great depths and that they deposited their load within short distances of the surface, often under conditions of increasing temperature.

W. H. Emmons.—I believe that the solutions that deposit the lode ores tend to rise to the tops, or highest points, of the batholiths and to spread out from the tops much as oil accumulates in domes. Some, of course, rise to noses and ridges and smaller cupolas on the sides of the batholiths. Where fractures provide channels, they deposit ores. Essentially all lode ores are features of the roofs of batholiths. The metals tend to rise to high points of the batholiths, except gold, which is more abundantly developed in the lowest points of the roof—that is, in the roof pendants.

Waldemar Lindgren.—Billingsly gave the most convincing proof that the ore deposits of the Montana batholith, except Butte, had their origin near the roof of that batholith and not in the depths of the batholith below.

W. H. Emmons.—The Butte batholith has an unusually flat-topped roof, and we find some puzzling things there. In fact, lead ores occur very close to the top, which is unusual for so large a mass.

J. C. Anderson.—The Mammoth mine north of Tucson is certainly in a batholith of considerable extent.

W. H. Emmons.—Some lodes are found within large batholiths, especially when there was a period of metallization later than the one attending the batholithic intrusion. That is probably the case with respect to the Idaho batholith described by Professor Lindgren. Even mercury ores, which generally are farthest out of all the series, are found in small amounts in the Idaho batholith. The deposits near Mammoth, Ariz., that is, the Copper Creek deposits, are in a granite mass that is less than 2 m. wide. It is an acrobatholithic ridge.

J. E. Spurr, New York, N. Y. (written discussion).—As this unusual paper purports to be a critical review of my views and an exposition of those of the writer (Lindgren), and as his presentation of my views is erroneous and misleading, my own comment will appropriately be preceded by a statement as to what, essentially, these views are. I have put them in tabloid form in an editorial,[10] as follows:

> The origin of gold-quartz veins as a variation of pegmatites has been argued as far back as 1861 by Thomas Belt, the distinguished English mining engineer and naturalist, but the idea was ignored by geologists. These views were independently

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[10] *Eng. & Min. Jnl.-Press* (1925), 874.

advanced by J. E. Spurr in 1898 as an explanation for the Yukon gold-quartz veins This conception, however, was of highly heated water as the end product of granitic differentiation, this magmatic water being the ore solution. But in 1905, studying similar gold-quartz veins in Nevada, Spurr pictured the ore solution in this case as concentrated and the gold-quartz as intrusive, just as pegmatites are intrusive. In 1907 the Zonal Theory of Spurr showed the interrelation of the different metals. This theory postulated that in the final pegmatitic residue of a magma many metals are present, and that with decreasing temperature each is separately precipitated, the deep-seated gold-quartz ores representing one of the deepest zones, close to the pegmatites. This theory, now generally accepted, pictured at first the ore solutions essentially as magmatic waters, except in the case of certain gold-quartz veins and ore-bearing pegmatites. The Ore Magma Theory, proposed in 1923, substituted for magmatic waters, highly concentrated and dense magmatic residues; for long circulation, a definite stage of injection; for fissures already open, the forcing open of fissures in many cases by the ore magma; for slow accretion of precipitation, simple crystallization from the injected ore magma.

For a longer presentation, I refer the reader to my presidential address to the Society of Economic Geologists.[11] In this paper I sketched briefly the development of magmatic theories of ore deposition, from the time of Elie de Beaumont, in 1847, and Lyell in 1857, through to the revival of this school in the United States. I wrote:

> But directly after Posepny's paper, a young American school arose, which, making use of field observations and consequent inductive reasoning, worked essentially back to the school of de Beaumont and Lyell . . . Of this school were Kemp, Weed, Lindgren and myself, and, later, many others; and many important papers, which I will not try to enumerate, built up an impressive array of facts and conclusions. The source of mineralizing waters was more and more referred to the igneous magmas, as de Beaumont and others, including the Austrian geologist, Suess, had done; in other words, such waters were not conceived of as surface-derived or atmospheric water, but as largely new water, born from the crystallizing igneous rocks.

This school of thought I called the "hot-spring school," to separate it from the "lateral-secretion school," which latter was predominant in the United States in the early part of my professional career, and which still survives in the Mississippi Valley and elsewhere.

In outlining the progress away from these schools, to which I and other American geologists at one time fully subscribed, I observed in the above-mentioned address:

> The two opposing schools of thought, which we may call the lateral-secretion school and the hot-spring school, both ascribe the deposition of ores to the agency of circulating waters, whether atmospheric or juvenile. Sharply defined from both these opposing theories is the theory of magmatic differentiation, first proposed, so far as I know, by J. H. L. Vogt, of Christiania, in 1893. The idea of magmatic differentiation of rocks dates as far back as Scrope in 1825.

And on page 624:[12]

> Vogt explained certain ores as due to this process of magmation, therein, as I said before, establishing a markedly different new school of thought in the explanation of

[11] J. E. Spurr, *Econ. Geol.*, 18 (1923), 617.
[12] J. E. Spurr, *Op. cit.*, 624.

ore deposition. Instead of water, whether atmospheric or expelled from cooling magmas, as hot springs or fumaroles, Vogt appealed to the igneous magma itself as the carrier and the segregator of ores; and thus water and magma as agents of ore concentration are clearly set off from each other. The question then comes up: What is magma? We know it is the fluid from which igneous rocks crystallize; but we now recognize it as also the source of certain concentrations of metallic minerals; and it is also the source of much water and of gases of many kinds . . . The magmatic segregations of Vogt are evidently considered as dry segregations, since he eliminates gaseous agencies, the chief of which is water; and he admits as magmatic segregations almost exclusively ores connected with basic rocks, considering the ores as among the first minerals to crystallize—according to the petrographers' conclusions, arising from the frequently clean-cut or idiomorphic form of these ore minerals, where they occur in the igneous rocks; and considering that these earliest-formed minerals have migrated, by a process which he does not satisfactorily explain, so as to concentrate. My own views as a self-styled magmationist, therefore, find no support in the views of the founder of the magmationist school. While Vogt found his inspiration in the extremely basic rocks, I have always been fascinated by the extremely siliceous igneous rocks—granites, quartz-feldspar rocks or alaskites, pegmatites, quartz dikes, and related quartz veins. Certainly, however, these siliceous extremes are as truly magmatic as the basic extremes; and when we recall that even in granites it is universally acknowledged that water and other gases are essential magmatic ingredients, without which there would be no granite magma, it is realized that Vogt has erred in excluding the agency of gases from the processes of magmatic differentiation.[13]

My own chief contributions to advanced thought along these lines may be marked by three stages, which are concisely summarized in the editorial quoted in the beginning of my comment. These were the three stages: First, the gold-quartz-pegmatite theory, which was original, but in which I years afterward discovered that I had been forestalled by Thomas Belt in 1861, and by him alone; this theory involved the recognition that certain gold-quartz veins were transitional from pegmatites (1898) and that they crystallized not from gradual precipitation from circulating hot waters, but from a concentrated gold-quartz magma (1905), which played the role of an independent intrusion; second, the conception and announcement of the zonal theory in 1907, which was original with me and was elaborated upon repeatedly in subsequent years—this theory postulating that "in the residual magma from which, for example, pegmatites and later gold-quartz veins were formed, the other metals were also contained, and that these passed into the magma still residual from the consolidation of the gold-quartz vein, and were successively deposited each at a separate and critical temperature of freezing, with the result that the different metals were ideally deposited one above the other, or one outside the other, with falling temperature, and thus ores of copper, zinc, lead, silver, and other metals were successively deposited;"[14] third, the ore magma theory (1923), which was radically different from all previous views, and which submitted evidence that

[13] J. E. Spurr, *Op. cit.*, 626.
[14] *Op. cit.*, 628.

not only gold-quartz veins and pegmatitic veins, but "many veins of copper, zinc, lead, and the like" have been "deposited from highly concentrated magmas, which appear in many cases to have intruded the rocks under pressure, much as does an igneous dike."[15] These relatively concentrated solutions I called *ore magmas* dividing them into relatively dry and relatively aqueous classes. Yet even the aqueous ore magmas:

> . . . according to my conclusions, are relatively highly concentrated metallic solutions, and are quite different from the mineralizing waters which I have hitherto pictured as the ore carriers in the case of copper, lead, zinc and such metals.[16]

At the same time I proposed the name *veindike* for certain ore deposits:[17]

> . . . for those veins such as have been described by Belt, Vogt and myself, as well as many others, which have been intruded as a thick magma, I propose the name veindikes, for all veins have not been so formed, some having originated from thinner although I believe generally highly concentrated solutions, and having been formed by replacement rather than intrusion.

I further stated that the ore magma of the dryer type "acts as an intrusive, and forms a veindike—a copper veindike, or copper vein, for example;" while the corresponding ore magma of the more aqueous type "acts more effectively by replacement and forms a replacement fissure vein, or an irregularly shaped replacement, or an impregnation, or dissemination."[18]

These three stages of progress of my geological thought were revolutionary, and were not followed by my old associates of the "hot-water" school. The present widespread recognition of the applicability of these progressive discoveries, as I think I may call them, has come mainly from a larger group of geologists, scattered world-wide, and with great field experience. The significance of the intrusive nature of certain gold-quartz veins, which I explained in 1898, 1905, and subsequently, was not grasped by the "hot-water" school; the zonal theory of 1907 was regarded by some of this school as positively heretical in its unorthodox views; and although it was quietly assimilated, not until 1922 did it become current thought in the universities, due to considerable part to its recognition by Professors Kemp, Rastall, and Emmons in 1922–1924. The ore magma theory, however, as expounded by myself in 1923, has had an immediate and world-wide assimilation, both with and without explicit recognition. Although of the old group of American "hot-water" geologists, of which I was also one, Lindgren has hitherto remained astonishingly immune from the influence of all these new ideas, as a study of his textbook shows.

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[15] *Op. cit.*, 631.
[16] *Op. cit.*, 632.
[17] *Op. cit.*, 631.
[18] *Op. cit.*, 632.

Summary of Spurr's Views

With this necessary preliminary, I will take up some of the points which Lindgren makes in the paper to which this is contributed as discussion. His introduction and resumé of the development of thought among the geologists of the magmatic group calls for no particular comment, but his "Summary of Spurr's Views" calls for correction. He states that I believe that "the majority of ore deposits are termed veindikes . . . in fact it appears that all ore deposits, whether of replacement origin or not are 'veindikes'." This is quite erroneous (as is shown in the above quotations from my writings), and shows a decided confusion of thought. I have defined veindikes clearly enough, and often enough. Replacement deposits are not veindikes; even replacement fissure veins are not veindikes. An intrusive vein is a veindike. Otherwise this passage does not call for discussion at this time. He closes by saying that fundamentally my views do not differ so greatly from those of other "magmatic" geologists, and quotes with hearty approval a summary in my paper proposing the zonal theory in 1907. I am indeed glad to know that Lindgren now finds only minor points on which to disagree; nevertheless, his textbook, in this case, shows no connection with his present frame of mind.

The Magma

Now for the minor points of difference. He does not approve of the term "ore magma." He has printed this passage (headed The Magma) before, in the proceedings of the National Academy of Sciences, and I have already answered his objections.[19]

Various distinguished geologists having attempted to define magma, with varying but on the average poor success, Prof. Waldemar Lindgren essays it in a paper read before the National Academy of Sciences.[20] He says:

"The word magma has a perfectly definite meaning. It describes a liquid of high temperature consisting of a solution of complex silicates in each other, perhaps with admixed oxides and always containing a certain quantity of dissolved volatile matter . . . I submit that logical usage demands that the term 'magma' be restricted to the above definition . . . When differentiation has proceeded to the point where only one component or several similar components remain, the term magma should be dropped. One may properly speak of quartz, or iron ores, or gases *of magmatic origin.*"

Considering this, the editor would first move to strike out the word *perfectly* as redundant and too jocund for the dignity of the National Academy of Sciences. It arouses the suspicion that Professor Lindgren may be spoofing, so that we examine the definition with greater care. Essentially, Professor Lindgren says a magma is a liquid. But at the high temperature which he mentions, it is universally acknowledged that the water or its components are gaseous, and that the other gaseous

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[19] J. E. Spurr: What is a Magma? *Eng. & Min. Jnl.-Press*, **120** (1925), 562.

[20] *Proc.* Nat. Acad. Sci., **11**, No. 1, (January, 1925).

components are varied and important. Therefore *fluid* would be a more accurate term than *liquid*, since liquid is opposed to gaseous. Passing over the wonder why in the perfectly definite definition the "admixed" oxides are prefaced by a "perhaps," one pauses at the dictum that the volatile matter is "dissolved" in the silicates. Why not, the freshman inquires, vice versa: to wit, the silicates dissolved in the gases? Or why not the silicates and the gases in part coexisting and separate? Indeed, the ancient and modern conception of granite magmas at least—a conception the general truth of which is well founded—is that the granite magma remains fluid at comparatively low temperatures by virtue of the volatile constituents.

Finally, omitting various other reflections for the sake of brevity, Professor Lindgren would surely not, on second thought, insist on the last-but-one sentence quoted. He would grant, for example, that the fluid from which a pyroxenite has crystallized is a magma, although pyroxenite by definition consists of pyroxene; or dunite, which by definition consists of olivine; or anorthosite, which by definition consists of the feldspar labradorite; or albitite, which by definition consists of the feldspar albite; or many other rocks. Indeed, it is a matter of common nomenclature that a magma may yield one mineral almost exclusively, or be "monomineralic." Daly observes, for example, ("Igneous Rocks," 1914, p. 447): "The gravitative differentiation of anorthosite . . . implies the segregation, in depth, of the monomineralic dunite magma, the bimineralic wehrlite or harzburgitic magmas, or the trimineralic lherzolitic magma." Professor Lindgren, now being supposed to retract his statement so far as it applies to monomineralic magmas crystallizing as olivine, or pyroxene or feldspar; should he be obstinate concerning a monomineralic magma crystallizing as quartz, claiming that it is no "magma?" Quartz is as truly and as undisputed a magmatic mineral as the others in question. Pegmatites are referred to by Daly as crystallizing "from gas-charged magma"[21] and Harker refers to pegmatites as representing, "the final residual magma of plutonic intrusions," and Lindgren himself says "the pegmatites are essentially residual magmas,"[22] a slight carelessness of diction which we may overlook. Also he speaks of "the transitions of pegmatite dikes to deep-seated ore-bearing veins."[23] That a phase of pegmatite consists of pure quartz is universally acknowledged.

Professor Lindgren also objects to the conception and term of a magma which crystallized as iron ores. Probably all authority and logic is opposed to him here. To quote from the handiest book for the moment, Daly refers[24] to titaniferous iron ores as "obviously local differentiates of gabbroid magma," and observes that "the sulphide ores of Sudbury are magmatic differentiates." And Professor Lindgren himself, in a paper by Lindgren and Davey on the ores of Key West, Nev., describing nickel ores containing magnetite, pyrite, and chalcopyrite, writes of the metallic minerals, "We believe that they formed part of the magma of the dike and consolidated as the last of the minerals of igneous origin."[25]

Indeed it would appear that Professor Lindgren's definition of a magma, instead of being "perfectly definite," is singularly inaccurate. The sharp line which he desires to draw between rocks containing metals in paying quantities (and therefore called "ores") and those that may contain similar metals, but not in paying quantities (and therefore called simply "rocks") is properly a commercial distinction and not a scientific one. The term "ore magma" introduced by Spurr is, as above shown, accurate; and, indeed, is so obvious that it can barely claim novelty.

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[21] Igneous Rocks and Their Origin (1914), 368.
[22] Mineral Deposits (1919), 762.
[23] Mineral Deposits (1919), 761.
[24] Igneous Rocks (1914), 454, 455.
[25] *Econ. Geol.*, **19** (1924), 309.

This question of what to call the child is not essential; still, I feel that Lindgren is captious and that his position is untenable. Sir Thomas Holland, president of the Institution of Mining and Metallurgy, recently[26] referred to certain pyrite ores as crystallized from a "hydropyritic magma." The term "ore magma" doubtless will stick: it differentiates ore solutions sharply from hot waters, and so serves an essential purpose.

"It is very difficult," says Lindgren, "to ascertain the composition of Spurr's ore magma." It is, indeed: I am still trying to get further light on this. It is also very difficult to ascertain the composition and condition of a rock magma. We should all investigate and study these problems patiently. Horace Freeman[27] has advanced an explanation which I am inclined to think throws light on the subject of ore magmas, but these magmas are evidently complex both as to composition and condition. Lindgren is perplexed because I described the ore magma as at times highly fluid and capable of replacement, and at other times gelatinous or viscous; but the same general observation is true of rock magmas and especially of pegmatite magmas. I may observe that first I divided magmas into three groups—rock magmas, pegmatite magmas and ore magmas. Subsequent study led me to join the last two together—the ore magmas are, I believe, best conceived of as a phase of pegmatite magmas. If one would get my idea as to the nature of ore magmas, let him consider pegmatite magmas: Every quality of the former is matched by the latter. The pegmatite-ore magmas (or pegmatore magmas, as I have called them) form in general a distinct group from the rock magmas, although there are transitional phases. Ordinary quartz-feldspar pegmatites, as it is well known, form dikes (veindikes), replacements, and disseminations, according to the variety. Ore magmas do the same.

The Differentiation of the Ore Solutions

Under the heading "the differentiation of the ore solutions," Lindgren is vague. It is to be pointed out that magmatic differentiation in the usual sense of the term—meaning the process whereby rocks split up into units of varying composition—must not be confounded with the loss of volatile constituents on sudden cooling. Magmatic differentiation, as Lindgren remarks (in accordance with what I point out in The Ore Magmas) does not begin until magmatic equilibrium is distributed by magma surgence, but a long static period of adjustment in a fluid magma is necessary for the development of important rock or ore differentiates. Even pegmatites are characteristic only of rocks which have consolidated under these conditions of very slowly changing equilibrium.

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[26] *Mining Magazine*, **34**, No. 3 (1926), p. 82.
[27] Horace Freeman, *Eng. & Min. Jnl.-Press*, **120** (1925), 973.

Dikes and Pegmatite Dikes

As to Lindgren's summary of "dikes" and "pegmatite dikes," it appears that we are in agreement, except as to his slightly different suggestion that he believes pegmatites normally result from the consolidation of acidic rocks "whether abysmal or intrusive near the surface." To this remark applies my observation on magmatic differentiation above. I do not think, indeed, that Lindgren would maintain this argument, on reflection. Pegmatites are characteristic not only of acidic rocks but of rocks of more femic composition, and they develop only if the time for differentiation is sufficient. Pegmatites, for example, do not develop from the great intrusive masses of quartz porphyries—these porphyries have been fluid after surgence for too brief a period.

Lindgren agrees with me as to magmatic pressure forcing open dike fissures.

Relation of Pegmatite Dikes and Veins

As to the relation of pegmatite dikes and quartz veins, Lindgren is still, as of yore, inclined to be conservative. He has not seen these transitional cases; I have. Yet what he does grant seems to amount to an agreement. "Some of these veins are evidently directly derived from intrusions by differentiation and so are the corresponding pegmatites." But he is inclined to think the two belong to "two separate successive stages." "It is clear that we have two things: 1, The pegmatite stage; and, 2, the hydrothermal stage, the latter containing much less alumina and more water than the first." To me it is equally clear that there is no such natural classification; certain pegmatitic magmas are assuredly aqueous; and the gradual transition from quartz-feldspar pegmatites to predominatingly or essentially pure quartz pegmatites is one of the commonest field phenomena that I know about. Certainly the pegmatites and the gold-quartz veins represent, by definition, two stages; but to call one "magmatic" and the other "hydrothermal" is simply to disregard field phenomena, on account of the old assumption that any ore deposit must be deposited by water.

As to the albite in gold-quartz veins, Lindgren says it is "not a magmatic phenomenon at all. It simply forms . . . under certain conditions, the reason for which we do not understand." But where it occurs in pegmatite dikes, as in those described by Schaller and Hess, Lindgren, on an earlier page, believes with me that albite with other minerals is due to "new supplies of magmatic juice of different composition from that of the original dike material," and like myself, disagrees with Schaller, who thinks they may result from later "hydrothermal solutions." And where albite occurs as practically the only constituent of dikes (albitite) carrying

(somewhat later) gold, as described by Turner and by Reid in California,[28] and which I have observed in Cuba,[29] it must be allowed that the albite is a "magmatic phenomenon." The significance of all these occurrences has not been sufficiently emphasized. Also I have pointed out[30] that in the case of the gold-quartz veins of California, as well as those of Ontario and Australia, where the feldspar of the quartz veins is sodic (albite), the alteration of the wall rock in each case has resulted in an increase of potash and a decrease of soda. The vein (veindike) solutions were therefore sodic; but those which altered the wall rock were potassic. "Evidently, therefore, the wall-rock alterations were accomplished, not by the vein-forming magma, but by solutions, gaseous and perhaps liquid, which were excluded from the gold-quartz magma on crystallization."

Neither can Lindgren's disposition of the adularia in Tonopah, for example, as due to country rocks rich in potassium, hold: for I have distinguished six distinct successive vein injections,[31] of which the fourth consisted of mixed quartz and adularia, whereas adularia is rare or lacking in the other stages; but they all lie in the same rocks, or rocks of equal potassic content.

Lindgren writes in the foregoing paper:

> Spurr's conception of the California gold-quartz veins is somewhat curious. He says, for instance,[32] that the quartz does not penetrate or silicify the country rock, therefore it must have been deposited by dry ore magmas; in the country rock the alteration is to iron carbonate. I thought everybody knew that the alteration of the country rock is intense and in places, as along the Mother Lode, very extensive. Also that siderite is not one of the alteration products, only mixed ankerite carbonates or calcite with a great deal of sericite, which indicates strong hydration of the country rock from the vein forming solutions.

Concerning this passage, I am forced to say that after much study I have no idea what was in Lindgren's mind when he wrote it. I have noted the alteration of the wall rock of the California gold-quartz veins in various places, emphasizing, as Lindgren has done, that while the veins are siliceous, the alteration of the wall rock has been to carbonates (of lime, magnesia, and iron) with sericite.[33] I am not clear whether some of the iron carbonate occurs as siderite. It does according to W. H. Emmons,[34] but this point is of no importance whatever. My first thought on reading this passage was that Lindgren meant that the wall rock has been silicified, but on checking up I find that he could not have meant this.

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[28] Cited by Lindgren, *Mineral Deposits* (1919), p. 570.
[29] J. E. Spurr: The Ore Magmas, **2** (1923), 613.
[30] J. E. Spurr, *Eng. & Min. Jnl.-Press,* **119** (1925), 890.
[31] J. E. Spurr, *Econ. Geol.,* **10** (1915), 751.
[32] J. E. Spurr: The Ore Magmas (1923), 523.
[33] J. E. Spurr, *Eng. & Min. Jnl.-Press,* **119** (1925), 890.
[34] W. H. Emmons: Principles of Economic Geology (1918), 237.

In an earlier writing[35] he observed "replacement by silica is not among the processes here recognized"—and a little later "it seems at first glance strange to see the quartz vein adjoined by a rock traversed in all directions by veinlets of carbonates." Now we know that rocks of all types are subject to silicification by penetrant solutions. These wall rocks have been penetrated by waters proceeding from the fissures, but they did not carry silica. On the other hand the adjoining veins, which according to Lindgren as well as myself, filled fissures and are not replacement deposits, are mainly of massive quartz. Surely this demonstrates beyond peradventure that the fluid which filled the veins was not water, but a fluid too dense to penetrate the wall rock—that it was an injected silica magma, from which was given off the water, carbon dioxide, and other materials which penetrated and altered the walls. This, it seems to me, is obvious.

The Fissures and the Filling

As to the open fissures, Lindgren agrees with me that they have in many cases been *filled* with vein material. He also agrees that they may have been held open by the pressure of solutions. He disagrees with me as to the manner of filling. He fails to see how an ore magma containing water and other volatiles could fill a fissure solid without constantly ascending supplies of fresh material. The answer to this conundrum is easy and is the same as in the case of a dike, which crystallized from a magma containing a certain percentage of volatiles—a percentage probably considerable in the case of granite and pegmatite dikes. These dikes, it is acknowledged, crystallized from a stagnant solution. The walls, kept open by the magmatic pressure, evidently closed in as the volatiles escaped and the bulk of the rock magma or pegmatite magma shrank. The same explanation applies to ore magmas. Gas-filled small residual spaces—vesicles in certain dikes, druses or vugs in pegmatite and ore veindikes—remain without pressure-closing, because they are isolated and thus strongly supported. In the case of many pegmatite and ore veindikes which are formed at considerable depths and in yielding wall rock, this closing-in pressure has brought the walls together, trapping the crystallizing magma into separate lenses, as I have shown in The Ore Magmas. I suspect that the application of this principle is broader than I have yet felt free to indicate.

Lindgren cites the Camp Bird vein as a case of deposition from moving waters: "The excellent plates of Ransome so convincingly show the gradual, in part colloidal, deposition from moving solutions that no further argument is necessary." But Ransome, in the same bulletin that Lindgren cites, says:

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[35] Waldemar Lindgren, 17th *Ann. Rep.* U. S. Geol. Surv., Pt. 2 (1895–6), 147.

In most of the important lodes of the Silverton quadrangle no crustification can be detected . . . On the contrary, crystallization has proceeded from many points within the solution . . . It is fair to assume that this structure in veins is the result of the undisturbed crystallization of a nearly motionless saturated solution.

I am in agreement with this.

Similarly, Lindgren cannot accept my belief that magmatic pyrite-carbonate veins and carbonate veins have been deposited from a saturated solution, which appears at one stage to have been gelatinous. He observes naively, "No physical chemist has yet been able to produce a stable carbonate gel." Neither has any physical chemist been able to make a tree; to recall Joyce Kilmer's poem,

Poems are made by fools like me
But only God can make a tree.

Nor have the physical chemists made a rock magma nor an orebody—whence one may apply Kilmer's poem to the present instance—

Geologists may write extravaganzas
But only God can make bonanzas.

Not that I see any necessity for a *stable* carbonate gel. R. A. Daly, it may be added, has recently described[36] carbonate (mostly calcite) dikes from the diamond mines of Africa, which he regards as essentially intrusive and magmatic.

Ore Magmas and Veindikes

A little later, Lindgren again misinterprets my views:

All ore deposits connected with magmatic emanations (except cinnabar and stibnite deposits) are evidently "veindikes." A careful study of Spurr's writings can lead to no other conclusion. The "ore magma" is differentiated from the parent magma, injected, consolidated. Nothing could be more simple.

Replacement or metasomatism is an awkward fact to face in this theory, and Spurr touches lightly on it. But this much is clear, that he regards pure replacement deposits like that of the United Verde mine, in Arizona, as "veindikes."

This interpretation finds no foundation in my writings. It is nonsensical to say that I regard replacement deposits as veindikes, or that replacement is an "awkward fact" for my theory.

But Lindgren would not object to calling some veins, veindikes:

Some magmas or fluids of magmatic origin may fill fissures close to the original source, without appreciable metasomatism. If it is desired to call these deposits veindikes, well and good . . . All this will make clear my reasons for rejecting the terms of "ore magmas" and of "veindikes" except as they may refer to certain pegmatite dikes and some very closely allied forms of quartz veins.

Here he evidently forgets that on a previous page he decided that a solution which crystallized as quartz alone could not be a magma.

[36] R. A. Daly, *Jnl. Geol.*, **33** (Oct.–Nov., 1925).

The Inclusions

Under "inclusions," Lindgren describes the inclusion, in veins, of wall rock or earlier-consolidated ore. He touches on the well known theories, including my own—my own being they are in many cases evidence of a relatively dense ore magma, not "circulating waters,"—and thinks all these theories may be applicable. He is not entirely convinced that such fragments have been brought up from great depths by ore magmas. I agree that this rarely occurs and the same observation holds for foreign rock inclusions in ordinary dikes.

The Ore Zones

Regarding my zonal theory and my interpretation of its various manifestations, in general Lindgren seems to approve. Quite properly he points out that the problem is a complex one, and that we do not yet know it all. He touches on the succession of crystallization in complex ores, stating that in general zinc blende antedates the copper minerals. He says:

> The succession is in places reversed or changed. Magnetite may follow pyrite and sphalerite may follow galena; sometimes pyrite and marcasite are repeated. Spurr believes this indicates rising temperature; it may be so, but this simple explanation is probably not the only solution of the problem.

I wish to make a correction here. I have advanced the explanation of rising temperature for the reversal of the normal zonal sequence in the case of successive vein injections, but I do not recall that I have done so for the crystallization of complex ores in a "telescoped" ore magma. I would not yet venture to do so. Conditions are different—the laws of eutectics, perhaps mass action, and many other complex physical and chemical laws, including those determined by variability of solvents, enter here. Freeman found that copper sulfide is fused with alkaline sulfides at a lower temperature than is lead or zinc sulfide, the order being the reversal of that indicated in nature by the zonal theory.[37] I could cite many instances of such complex crystallization phenomena, but I believe that where there is a chance of more gradual crystallization and differentiation, even in a complex ore, the normal succession called for by my zonal theory may be most often found—copper minerals preceding those of zinc. Thus in a paper[38] just come to hand, E. S. Moore finds the following succession in a lead-zinc deposit in Canada: pyrite, pyrrhotite, chalcopyrite, and either galena or sphalerite.

Lindgren refers to my observations at Angangueo, in Mexico, where there is a striking succession of great veins, successively injected in successively opened fissures. From a field study of these veins and

[37] J. E. Spurr, *Eng. & Min. Jnl.-Press*, **120** (Dec. 19, 1925), p. 977.
[38] E. S. Moore, *Bull.* Can. Inst. Min. & Met. (1926), 371.

their relative ages, I determined the general sequence[39] as: 1, pyrite and cupriferous pyrite; 2, blende; 3, galena (argentiferous); 4, silver-bearing quartz; 5, manganese carbonate. Many transitional phases occur. Lindgren proceeds that "the exact succession, as determined by Newhouse," shows galena and sphalerite preceding chalcopyrite. Looking up this reference, I was dumbfounded to find that this evidence, accepted by Lindgren as final, rests not upon any field work but upon the microscopic examination of some specimens sent to the Massachusetts Institute of Technology. Newhouse observes:[40] "No geological description of the deposit from which they were taken is available." Of the different minerals he says, "Little age difference is apparent." Apparently the specimen represented the crystallization of a complex ore. I have no reason to doubt Newhouse's observation, but surely they are only an inconsiderable item in the problem of ore sequence in this district and quite subordinate to the larger relations; and to cite them as conclusive, as Lindgren does, ignoring field relations, is not judicial.

In the next paragraph Lindgren states that I believe that the bonanza ores in general "are deposited under conditions of increasing temperature." I do not believe this. I have never set down such conclusions.

In the next paragraph he cites me as believing (as I do) that cinnabar and stibnite may form hot-spring waters and implies that I do not believe that gold may be deposited thus. But I have always specifically included a certain group of gold ores in this category, and described its association with cinnabar and realgar, from 1895[41] on. In 1925 I wrote:[42]

> In The Ore Magmas I have concluded that a certain group of metals have been dissolved by waters out from the ore magmas proper and thus derived from them. These theoretically water-soluble metal sulfides I reasoned to be arsenic, antimony, mercury and gold; essentially the group defined chemically as water-soluble by Freeman's experiments.

In the next paragraph Lindgren states that I believe that "all ore deposits of magmatic affiliation have been deposited at minimum temperatures of 365° C., and ordinarily about 400° to 500° C." This statement is quite incorrect. I have assumed the range 575° down to "perhaps 365°"[43] for the true ore magmas—not for the group which I have mentioned in my last paragraph above, nor for any minor solution and deposition which hot springs or gases may otherwise have. Lindgren proceeds: "I will simply say that if this were correct, all deposits in limestone would have a gangue of lime-iron silicates." But a little way

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[39] J. E. Spurr: The Ore Magmas, **1** (1923), 272.
[40] W. H. Newhouse, *Econ. Geol.*, 20 (1925), 56.
[41] J. E. Spurr, *Op. cit.*, **2** (1923), 699–705 et seq.
[42] J. E. Spurr, *Eng. & Min. Jnl.-Press*, **120** (1925), 976.
[43] J. E. Spurr: The Ore Magmas, **2** (1923), 834, 836.

ahead, under "Contact Metamorphism," he observes contradictorially, even if more intelligibly: "The only condition needed is that the temperature shall have exceeded 500° C., below which limit the lime silicates can no longer form."

Contact Metamorphism

Lindgren here recalls my conclusion that the so-called contact metamorphic deposits are, in all cases which I have seen, simply fissure veins or replacement pipes located somewhere near an igneous contact with limestone, and associated more or less closely with lime silicates, which are mostly older than the ores and mostly have, moreover, no definite space relations with them. In this at last he cites me correctly. In every case of this type which I have seen there is evidence that the sulfide solutions (ore magmas) rose from some considerably greater depth and were not expelled from the adjacent igneous rocks now exposed. The general distribution of these orebodies, indeed, completely negatives any idea that they result from universal magmatic emanations. If they did we should only have to have an igneous contact to have an orebody. But although there are many thousands of miles of igneous contacts, ore deposits are very rare. Moreover, where ore deposits do occur near an igneous contact, long stretches of the same contact are quite barren; and the ore is often notably associated with a small body of igneous rock, while more massive portions nearby have no associated ore. In fact, no practical prospecting rules have yet been found that would establish a reliable relation between igneous contacts and ore deposits. Apparently there is no fixed relation, economically, nor, indeed, genetically. I see in these orebodies only a variety of orebodies in general and their formation from a highly specialized and relatively concentrated type of ore solution (ore magma), which rose from the depths below, from a horizon where not only consolidation but magma differentiation could and did take place. My demonstration of the relations of such orebodies to associated igneous rocks, convincing in their nature, caused Lindgren to admit that this relation is true in some cases; nevertheless, he believes that there is "such a mass of cases where no such deep-seated fissures can be postulated, that it simply seems absurd to me when the actuality of emanations from adjacent intrusions is denied." I do not deny it. I have been looking for such cases and have never found them; and the cases that Lindgren cites in confirmation of his views seem to me to support mine. For example, he cites White Knob in Idaho, where "the alteration has gone so far that limestone and igneous rock can no longer be sharply differentiated." Certainly in this case he cannot claim that the solutions originated in the igneous rock at or near this horizon and passed out into the limestone: a far deeper source for the solutions which have altered both is demonstrated. Secondly, he cites the map of

Copper Mountain by Wright: "I cannot find anywhere a more convincing proof of my views." But Wright[44] reports that "dikes or veins" of the lime-silicate rock "branch out into the limestones and schists and occupy fissures in the granite intrusive."

Are not these fissure veins? Elsewhere the ore, Wright says, "grades into the intrusive mass." This zone of gradual replacement is often many feet wide. There is also a gradual change of the intruded schists, due to replacement. The progress of mineralization is shown by Wright to have been marked by a gradual sequence of events:[45] 1, alteration of limestone and *intrusive diorite* to lime silicates—especially garnet and pyroxene; 2, replacement of calcite by chalcopyrite and pyrrhotite; 3, *filling of fissures* with pyrite, chalcopyrite, magnetite, and other minerals; 4, deposition of zeolite and calcite. There are also associated quartz veins belonging to the later stages of this same sequence.

Altogether the phenomena clearly disprove Lindgren's hypothesis of the derivation of the ores from the igneous rock exposed; these ores must have come from far greater depths, and along fissures (as Wright states[45]) at a period distinctly subsequent to the intrusion of the exposed rock and its cooling.

Conclusion

On examination of this paragraph of Lindgren's paper, as well as his preceding pages, I am impressed with the fact that he has half adopted many of my conclusions. The term "ore magma" is rejected except for "special cases." The term is therefore allowable. Ore solutions may keep fissures open by their own pressure. "The filling was deposited gradually by mobile solutions, not from thick and viscous fluids, though close to the magma the viscosity no doubt was higher than nearer to the surface." He grants, therefore, the occasional viscosity of ore magmas, which is almost as far as I go myself. He makes no mention of "circulating waters," or of "hot-spring waters," according to the pre-ore-magma thought. In a recent paper he even says of a specific case that "there has been an *injection* of highly mobile aqueous solutions."[46] I do not find the time-honored phrase "attenuated solutions" in his paper, a phrase and implication against which I have repeatedly protested; and the banner of the hot-water spring geologists—the word "hydrothermal" (meaning, pertaining to hot water)—is used sparingly. Evidently it also is undergoing obsolescence, although it may be applicable to such deposits as those of cinnabar. It is natural that during the 3 years since The Ore Magmas and my supplementary writings began to appear, there should

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[44] C. W. Wright: Geology and Ore Deposits of Copper Mountain and Kasaan Peninsula, Alaska. U. S. Geol. Survey, *Prof. Paper* 87, (1915), 45.

[45] C. W. Wright, *Op. cit.*, 108.

[46] Waldemar Lindgren, *Econ. Geol.*, **31** (1926), 144.

have been much assimilation, perhaps partly subconscious, even in the minds of many geologists who have been loath to change their earlier conceptions.

Lindgren emphasizes the great importance of replacement in vein formation. He will find nobody to disagree with him as to this ancient truth. But he does not believe that coarse-grained quartz can be formed by replacement of aluminous rocks. I believe that it may be possible but that many cases which have been ascribed to replacement are due to fissure-filling, as Lindgren does. He rejects the term "veindike," "except for certain pegmatite dikes and other exudation products of purely magmatic origin." That is, he accepts the term as a term. Lindgren grants the existence of unsupported inclusions of country rock in veins, but thinks that in some cases they fell from the walls and were supported by the partly filled vein. As many of these veins show no banding, the partly filled vein must have been a dense fluid, to support them. He decides that he has examined the zonal theory and that in general it is true, but that there are many irregularities—which I have long ago shown. As to the so-called contact metamorphic ore deposits, he does not deny that some may be due to solutions ascending from deep sources.

Altogether, critical analysis would indicate that the effect of this paper is with reservations, to acknowledge acquiescence to the majority of my revolutionary views, even though his very contentious manner seems to be intended to give the impression that he is protesting; vowing he will ne'er consent, he is consenting.

There remain, however, certain points where a real divergence of views is shown. He does not think there is an unbroken transition from pegmatites to gold-quartz veins. I am sure that there is, and there are many who, having observed this, agree with me. He does not believe that the presence of albite and adularia in veins indicates a relationship to pegmatites. I believe that it often does, especially in the case of albite; in the case of adularia my view is not so well advanced, but it may be so. He does not believe that floating inclusions are "carried up" by vein-forming solutions. I have at different times submitted varied and convincing evidence that these floating inclusions have been transported for varying distances by the ore solutions (ore magmas). He still believes that lime-iron silicates and sulfides may be "a universal phenomenon attending the solidification of magmas." Unfortunately for the prospector and miner they, in point of fact, are not. Emanation of water and other gases from consolidating magmas is a universal phenomenon. But the very fact that orebodies are rare, not universal, constitutes by itself a sufficient proof that these exuded waters and gases are not the ore solutions. By assuming this false identity, marginal ore deposits near igneous intrusives are confused with contact metamorphism caused by these magmatic exhalations. In the study of ore deposits by Lindgren,

wall-rock alteration, local or widespread, which is usually due to widespread suffusion by aqueous magmatic exhalations, is confused with ore injection, which in most cases is local and sharply defined (in point of time or place, or both) from such alterations. The two phenomena—one common and widespread and the other rare and highly localized (the first the alteration of rocks, whether or not near igneous contacts, and clearly due to hot magmatic waters or gases; the second, ore injection, which is, wherever it occurs, clearly due to concentrated and highly localized magmatic differentiates—ore magmas) must be recognized as essentially distinct, before we can proceed further with the interpretation of ore deposits. The former (hot water, or gases) are given off from cooling magmas, wherever they occur; but the ore magmas, like other magmatic differentiates, only collect together where conditions are favorable for this and other manifestations of magmatic differentiation, in what I have called the zone of differentiation—a zone whose depth is variable.

Finally, I regret that Lindgren appears to be completely oblivious to what I regard as some of the most important and constructive points which I have brought out in my writings during the last few years; for he does not mention them, either pro or con.

And, as the very last word, I must say that as yet we know nothing as to ore deposits compared with what there is yet to learn. Many clues invite us to further investigation; let us follow them up—that the science may move forward.

WALDEMAR LINDGREN.—I am very glad indeed that Mr. Spurr has called attention to one point regarding which I have quite unintentionally misinterpreted his views. For this misinterpretation I apologize. It refers to his definition of a "veindike." It is very difficult to find clear statements about the relation of "veindikes" to replacement deposits in his book on Ore Magmas, although in some of his later articles the distinction is more definitely outlined. When, for instance, it was stated that at Jerome "the basic dikes—were contemporaneous with the final phase of the copper magma injection."[47] I assumed that this copper magma was a viscous differentiate. I realize now that Spurr restricts the use of veindike to what may be called "filled fissures," and strictly separates replacement from such injection. As I interpret his views, the replacement ores were formed by more mobile solutions, which either separated from the viscous magma of the vein dikes itself, or came up as independent injections of ore magma of tenuous consistency. We cannot understand Spurr unless we fully realize that an ore magma may be either viscous (or gelatinous) or liquid, or gaseous.

Now that this misconception is cleared up (again with my apologies) let us apply this veindike definition to various deposits.

[47] J. E. Spurr, *Bull.* Geol. Soc. America, **36** (1925), 559.

Spurr states that the Camp Bird deposit is a veindike, also the silver deposits of Cobalt and the filled veins of the Georgetown district, Colorado. On the other hand, he states that the Kirkland Lake gold deposits are *not* veindikes, and that the Hollinger gold deposit is about 50 per cent. veindike and 50 per cent. replacement ore. The Braden and Jerome copper deposits are not veindikes, being almost wholly replacement ores. The California gold deposits would be veindikes in part. Now when we consider that it is most difficult in many deposits to say just what is replacement, and what is filling, we realize that all that has been accomplished by this distinction is to substitute for the general term "vein," which does not commit us to any given theory, the partial term "veindike," which commits us to the magmatic theory of injection of (I suppose) a viscous ore magma, and which throws together such widely separated things as pegmatites and deposits with delicate and repeated crustification following the walls. It would be interesting to ascertain, for instance, what proportion of the Butte deposits is "veindikes," and what is replacement deposits. I should guess that Mr. Spurr would call the Rainbow lode a veindike and the veins in the Leonard Mine, in the copper belt, replacement deposits. It would seem to me that this discussion fully confirms the conclusion in my paper, and makes it more emphatic, namely, that "veindike" is a wholly undesirable term in our modern vocabulary of ore deposits, except in the limited way indicated in my paper (p. 82).

I have referred sufficiently in my original paper to the great changes which take place in the upward course of ore solutions after having effected replacement.

Concerning the historical part of Mr. Spurr's rejoinder I have little to say. Probably authors are not the best of historians, particularly when it comes to relating their own discoveries. I do object, however, and most vigorously, to be classified as belonging to what Mr. Spurr calls the "the hot-springs" school. This was represented, and most ably by Stelzner and Posepny. Clearly and unmistakably I stated my position in 1907,[48] and in 1906,[49] before Mr. Spurr's magmatic theory saw the light. The incentive which directed many of us in these new channels was the work of Professor J. H. L. Vogt.

I have cut out the reference to the Angangueo deposits, discussed by Mr. Spurr in his rejoinder; it seemed not essential to the subject and was mainly introduced to show that the sequence of mineralization of any district cannot be correctly interpreted except by the aid of all

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[48] Waldemar Lindgren: Present Tendencies in the Study of Ore Deposits. *Econ. Geology*, **2** (1907), 743–760.

[49] W. Lindgren and F. L. Ransome: Geology and Gold Deposits of the Cripple Creek District, Colorado. U. S. Geol. Survey, *Prof. Paper* 54, (1906), 217–231.

modern methods. Incidentally, Mr. Spurr misinterprets Dr. Newhouse's meaning.

I venture to suggest to Mr. Spurr a re-examination of the Aspen ores. I believe he will find there has been no reversal of deposition owing to increasing temperature. I believe that he will find that the "polybasite" (really the pearceite and the stromeyerite) is the latest of the series of minerals and that it replaces the galena.

It is indeed hopeful to think of this larger group of geologists—men of "great field experience." I am sure that they will accomplish much. But they will accomplish more if, after dictating the history of an ore deposit to the stenograper while standing on the outcrop, they will sit for a while at the feet of Physical Chemistry and apply for information to the instrument known as the metallographic microscope.

To conclude, I express again my high regard for Mr. Spurr's work. The difference between us is that he goes further in a certain direction than I care to follow.

From days of old, the study of ore deposits has been suffering from a superabundance of theories. Would it not be wise if all of us ceased for a while to promulgate theories and instead "got down to brass tacks?" If we have the facts, the theories will take care of themselves.

C. A. Porter, Philadelphia, Pa. (Written Discussion).—The question or unsupported inclusions of wall rock in veins, from the discussion following Lindgren's paper, seems to have gained considerable importance. Along with this, the main paper has touched on the difficulty of accounting for open fissures that have been filled in the deeper (hypothermal) ore zones. It is believed that definite answers can be given to these questions, and as the solution of the first question partly depends on that of the second, they may well be considered together.

Filling of Fissures at Great Depths

It has evidently been customary to consider that the openings of fissures have attained their full magnitude before the beginning of their mineralization. This may not necessarily be true; in fact, much evidence may be adduced that the openings in many fissures are of slow growth, and that the deposition of the ore and the expansion of the fissure may progress simultaneously.

In order to understand this, let us consider some facts that must have come under the observation of every engineer. It is doubtful if fissures ever exist, whose walls form perfect planes without swells or depressions, bulges or changes in the dip or direction along the strike. When a piece of crockery is broken, we all know that if a tight joint is to be made, any two pieces must be placed together in their original position. In the case of fissures, the unbalanced stresses that

cause the fractures preclude any possibility of the walls returning to their original position, and the bulges, when they are forced away from their corresponding depression in the opposite wall, if they are not ground to pieces in the movement, must leave the less prominent or depressed parts with open spaces between them.

Fig. 1 shows how a fissure may be opened gradually over a considerable period of time,[50] possibly taking place coincidentally with the mineralization, and even continuing after the mineralization has ceased, leaving open spaces, especially along the walls, as at Rico, Colo.[51] Striations upon vein fillings and upon walls are common and many faulted veins attest that movement may continue after the mineralization has begun and even finished. An important instance, in view of this discussion is

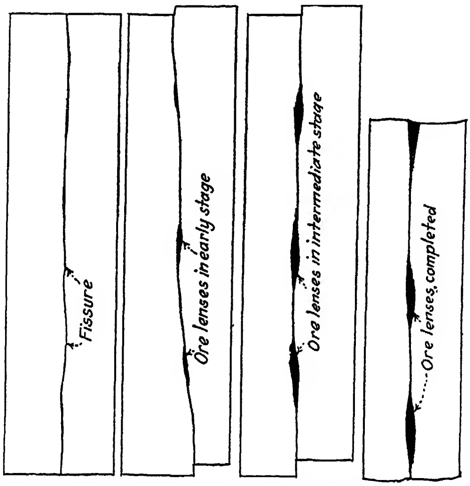

FIG. 1.—CROSS-SECTIONS SHOWING PROCESS BY WHICH OPENINGS MAY DEVELOP.

reported in the Amethyst vein at Creede, Colo.[52] "At this place the slickensided plane cuts through the ore." Obviously, as a fissure opens gradually and fills with gangue, at any given period only a small space is left open. Thus a maximum resistance is offered to pressure tending to close the opening. This, it seems, explains how open spaces could have remained open at great depths until they were filled with ore.

INCLUSIONS IN FISSURE VEINS

An understanding of the principle of gradually opening fissures helps

[50] C. A. Porter: Minor Crustal Movements and Ore Deposits. *Eng. & Min. Jnl.-Press* (1924) **118**, 782.

[51] T. A. Rickard: Vein Walls. *Trans.* (1896) **26**, 193. Cited in *Ore Deposits* (A. I. M. E., 1913), 226.

[52] W. H. Emmons and E. S. Larsen: Geology and Ore Deposits of the Creede District, Colo. U. S. G. S. *Bull.* 718, (1923), 163.

in the explanation of inclusions in fissure veins. Suppose a fracture has been formed, having the same dip as the vein in Fig. 2, that a 2-in. open space has resulted, and that solutions enter the space and fill it with ore and gangue. Another movement follows, forming a further space of say 1 in. between the hanging wall and the vein filling already deposited. A fragment of comparatively flat rock in the hanging wall becomes loosened, and falls forward until it rests upon the gangue deposited after the first opening. Such a fragment, it will be noted, has no opportunity to turn over, and its axes must remain in the same general position that they had originally in the wall. Now, after another accession of mineralizing solutions and a deposition of gangue, the loose rock will become enveloped in this freshly deposited vein filling and be held rigidly in

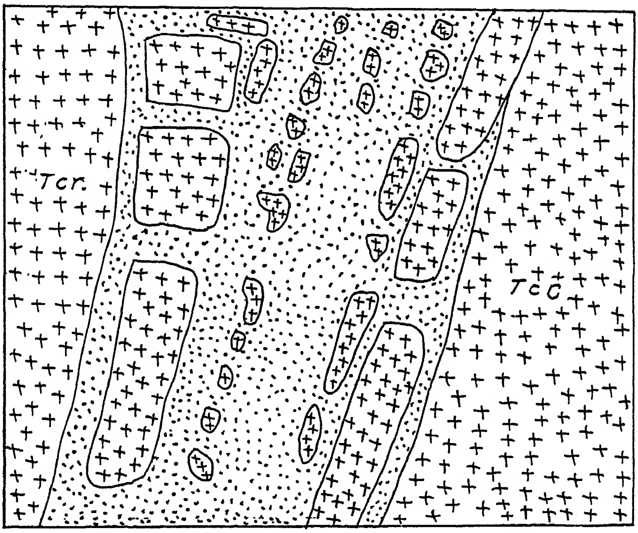

FIG. 2.—CROSS-SECTION OF THE OVERHOLT VEIN, CREEDE, COLO.

place. It already rests upon an earlier deposit 2 in. thick, and is thus separated from the footwall. If then a third movement takes place, opening the fissure again on the hanging-wall side, a third deposition of vein filling will then be formed between the encased fragment and the hanging wall, leaving the piece of rock separated from both walls, and supported entirely by the vein filling. The result is a true inclusion. Fig. 2, showing inclusions in the Overholt vein, Creede, Colo., illustrates this principle. The process described needs but to be repeated often enough to produce the condition shown.

OBJECTIONS TO VEINDIKE THEORY OF INCLUSIONS

Attempts have been made to explain the occurrence of inclusions by "veindikes." It has been assumed that a veindike, having a specific

gravity about equal to that of an inclusion, will cause it to float. No doubt such a condition is conceivable, but, it seems extremely improbable. However, assuming that such might be the case, the alignment of the inclusions, both with themselves and with the walls of the vein, are totally unaccounted for by the veindike theory. How a fluid, primarily in motion, and subject to all the changes of a still active magma (such as fresh impulses from below, movement of the walls and changes in temperature) could fail to turn such delicately balanced inclusions topsy-turvy, seems entirely beyond conception. Perhaps the greatest difficulty lies in the parallelism of the inclusions with the walls. In the veindike theory, there appears to be two forces at work, namely, gravity tending to pull the inclusions downward and the sustaining power of the veindike fluid. These two forces act vertically, and just how they could have

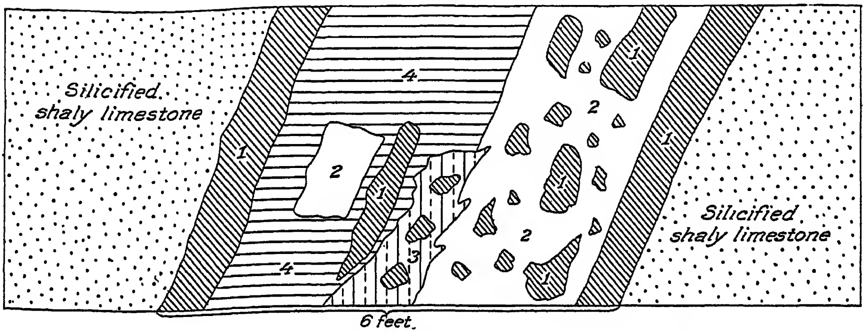

FIG. 3.—CROSS-SECTION OF VEIN AT PENOLES, DURANGO, MEXICO.

arranged the inclusions so orderly, about 20° from the vertical, certainly demands a specific and detailed explanation.

In Fig. 3, the difficulties are not lessened. Three different vein fillings all support inclusions, and in one place the same vein filling contains two different kinds of inclusions, but, in all cases, the veindike must have had the right specific gravity to maintain them until the intrusive fluid had solidified. Again we find a parallelism with the walls, although less marked. Also each type of vein filling attests a period of movement and disturbance, yet we must assume three periods of quiescence, the breaking of which for a second before the veindike solidified would have destroyed all such regularity. In this case the vein filling is of low gravity, there being no heavy minerals, such as blende, to offset whatever water and gas that must have existed to some extent in the original veindike material, hence it is hard to understand how they could have been heavy enough to support the inclusions.

Fig. 4, a cross-section of the Phillips vein, Georgetown, has been cited as an example in support of inclusions by veindikes. Apparently, there is but one inclusion, the strip of mixed quartz and crushed, silicified

gneiss, *c*, that extends up and down the middle of the vein. Witho analyzing the mechanical difficulties to applying the veindike theo here, which involve another set of phenomenal balances somewh differing from those already touched on, let us see if we cannot furnish understandable and quite simple explanation based on the principl already advanced.

Let us assume that the strip of ore on the footwall was deposit before that on the hanging wall, although it should be noted that t hanging wall strip could actually have been deposited first. The begi ning would be marked by the opening of the fissure through movement the walls in accordance with Fig. 1, this continuing until a space had be opened as wide as the lower strip of ore. Mineralization proceed either with the progress of the opening or soon afterward, the space w

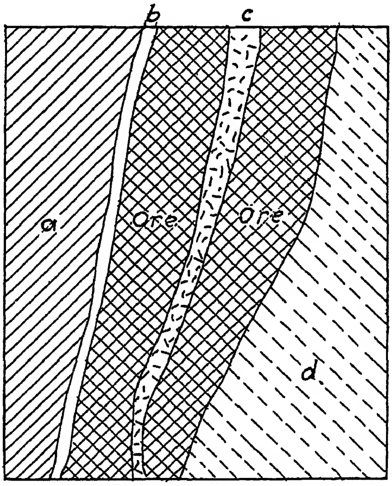

FIG. 4.—CROSS-SECTION OF THE PHILLIPS VEIN, GEORGETOWN, COLO.

filled and the lower strip of ore completed. Later the opening mo continued, but the hanging wall having become "frozen" to the ve filling through the deposition of silica or other gangue material, a stri was left attached to the orebody; a result likely also to be aided by an weakness due to slips in the hanging wall parallel to the original fissur The second movement then developed another open space above the o and above the "frozen" strip of former wall rock. This, in turn w filled with ore, leaving the strip of wall rock embedded between the fir and second deposits. The result is a long inclusion similar to that show in Fig. 4.

BLAMEY STEVENS, Real de Arriba, via Toluca, Mex. (Writte discussion).—Many passages on the emanation theory in Profess Lindgren's book, "Ore Deposits," appear to the writer much more co

vincing than the paper under discussion. The book appeals more to what the average reader knows from his own experience and less to rare phenomena, which are generally wrongly interpreted or wrongly or insufficiently described. The writer may therefore be permitted to describe the process of emanation in a generalized way, bringing the fundamental sciences to his aid.

The Principal Intrusive

The reason for the intrusion of a batholith seems to be as follows: Excessive erosion takes a load off some part of the earth's surface. Then deep-seated magmas begin to drift horizontally toward this area. These then bulge up the overlying rocks so that still more erosion is induced and so the cycle proceeds. The normal batholith is elongated and may be hundred of miles long. After it has risen well into the surrounding solid rocks, it continues to widen by pushing them apart horizontally. By the aid of water and other volatiles, schists and gneisses are thus formed near the intrusive and, farther away, folding of the horizontal strata takes place. The stress differences required are not very great.

Pegmatites

As the batholith cools, various minerals are crystallized out and form a "honeycomb" whose interstices are filled with the still liquid part of the magma. The cooling also brings about a general contraction, so that the stress on the walls of the batholith is relieved and they become subject to the play of minor forces. Thus some of the liquid phase of the consolidating batholith may find its way into the surrounding country. Liquid displaced from acid magmas may form veins or dikes of very acid rocks. These can be generalized under the term "pegmatites." Our fathers preferred to call them veins, because they usually have irregular forms.

These pegmatites contained more water and other volatiles than the parent magmas, but not enough to stop them from consolidating as virtual eutectics or end products of crystallization, as the structure of many of them distinctly indicates.

Water in Magmas

Barus[53] long ago determined by experiment that glass and water are miscible in all proportions at 200° C., and it follows that they must remain miscible at higher temperatures, provided the pressure is sufficient to retain the water at a density not too different from that of the Barus experiments.

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[53] Barus: Remarks on Colloidal Glass. *Am. Jnl. Sci.*, 4th ser. (1898) **6**, 270.

The discovery of this fact[54] was the key to solving the problem of ore deposits. The scientists of former generations had investigated every phase of the problem with remarkable insight, but they did not think that water could be dissolved in a red hot igneous magma, and so they had to be content with the circulation theory. Thomas Belt and other geologists intimately acquainted with ore veins had realized however that circulation was unable to account for the facts, hence Belt's own theory.

Point in Dispute

Just here is where the classic geologists, represented in this discussion by Lindgren, part company from Spurr whose theory is that the ore deposits are formed from the only slightly aqueous pegmatites. The classic geologists contend that the great majority of acid ore deposits are products of further cooling in the main igneous rocks, from which a very fluid aqueous solution is finally ejected, rich in the valuable metals. In detail the process is as follows.

Origin and Action of Emanations

As cooling further proceeded in the batholith, the interstitial solution became richer and richer in water and volatiles until the solution had expanded considerably. This expansion was also noted by Barus from experiments. The result of the expansion was to increase the pressure of the interstitial fluid and this increased pressure shrinks the material forming the "honeycomb" or solid phase of the cooling magma. This "honeycomb" cannot shrink as a whole, being prevented from so doing by the walls and the lower part of the intrusion. Therefore, it cracks or forms joints. The writer speaks with considerable confidence of this process of shrinkage, cracking because using a precise method[55] he has worked out the parallel case of vertical jointing in stratified rocks.

As a result of the jointing and the high interstitial pressure, the solution is squeezed out of the top part of the batholith, and it then takes the name "*residual emanation*," or simply "residue," if it is understood to be an emanation. As times goes on the process is repeated below, the jointing formed above being extended down and continuing as channels for the exit of the emanation. The top of the batholith is continually conveying these hot fluids, hence it does not perceptibly cool below their temperature. Thus, the emanation in passing through them deposits nothing. Also, as the emanation has been formed in contact and chemical equilibrium with similar material, no alteration or replacement

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[54] C. R. Van Hise: Principles of North American Pre- Cambrian Geology, (1894–95). *16th Ann. Rep.* U. S. G. S. Pt. 1, (1898) 689.

[55] Blamey Stevens: The Laws of Jointing. *Trans.* (1913) **47**, 91.

goes on within the batholith itself. We are, of course, considering a simple case, and not a complex one.

The process of emanation here described is definite, and the residue is distinct from the magma of the batholith. A homely example would be the coagulation of milk into curds and whey. The whey is a distinct substance from the milk, and if we had no word "whey," we should say "milk juice," or something of that sort. Similarly we might say "magma juice," but the word "emanation" is well established and the theory in general is now classic. Some sponsors of the theory may differ from the writer in details, and the final agreement, when arrived at, may differ in still other details, but in the main the emanation theory of ore deposits must surely stand.

Mineralization by Emanations

Above the top of the batholith the emanation may find faults or other ready-made fractures by which to escape, but often, and perhaps more generally, it opens narrow intrusive rents for itself, through which it proceeds toward the surface. Whatever the nature of the fracture, the solution, as it proceeds upward, gradually loses temperature and pressure and in doing so deposits its mineral content in a certain general sequence.

In the course of time the intrusive rent fills up with these deposits, and would choke itself off except that the partial stoppage leads to the accumulation of additional solution pressure, which pushes the walls further apart. This process of view-widening may be repeated each new break taking a slightly different course. Inclusions of country in the vein rock are no doubt often formed in this way.

Another important evidence of emanations is provided by hot springs. Records of some of these springs date back for thousands of years, and although the flow may occasionally fluctuate in such manner as would correspond with the readjustments just mentioned, these springs run steadily and plainly tell us they that have a continuous source of supply. The possibility of some of these springs being partly due to circulating water is not overlooked, but their mineral content cannot often be explained.

Another kind of emanation that acts very differently accompanies volcanism. It comes from magmas that have approached the surface so closely that they boil. These "exhalations" are spasmodic and not continuous. Theoretical considerations make it improbable that they can form at more than 20,000 to 30,000 ft. depth. Probably the average, exhalation does not come from more than half that depth. Therefore they hardly come into the present discussion.

The economic deposits that have originated in basic magma are largely replacements and have therefore been carried by water or other volatiles,

but the physics of the consolidation of basic magmas is less well understood than that of the acid magmas.

The use of the terms "differentiation" and "segregation" often implies the idea of a liquid silicate being segregated into two or more different liquid masses or magmas. Such a process is not acceptable on fundamental considerations and it seems, to the writer, to be better to avoid these terms, using instead such words as "emanations" "residue" and "fractional crystallization." Vogt, the most persistent exponent of the magmatic genesis of many basic ore deposits, recognizes the probability of such deposits being fractional crystallizations.

The idea of gels, or of any other non-fluid substance, flowing in veins or dikes hundreds or thousands of times as long as broad, is untenable. The pressure fall with such a substance is exponential to the distance, that is to say that as the distance increases in arithmetic progression, the pressure decreases in geometric progression and therefore soon becomes ineffective in moving the substance.

As the writer views the matter under discussion we have in the emanation theory an acceptable explanation of acidic ore deposits, one that can be founded on the fundamental sciences and supported by existing experimental evidence. He considers, that until the "ore magma" theory can be similarly sustained, it cannot expect general acceptance.

Intrusive Character of Ore-bearing Solutions

The writer hopes that Mr. Spurr does not wish to intimate that he was the first to recognize the intrusive character of most economic ore-bearing solutions. By "intrusive" geologists have always meant a fluid that makes room for itself by pushing its boundaries apart. Water simply flowing in a quartz vein or an iron pipe is not intrusive even though its pressure is very high. The writer has no copies of Spurr's Geological Survey papers, but any such conviction should decidedly have been brought out in his Institute papers as these dealt with deposits in which the solutions were intrusive.[56]

The writer believes that his paper[57] of 1911 and a somewhat later article in the Mining Magazine of London were for some years the only consistent contentions that many varieties of ore deposits have been formed from solutions that intruded the country rock.

[56] J. E. Spurr: A Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores. *Trans.* (1902) **33,** 288; *Ore Deposits* (A. I. M. E., 1913), 251.

——— Genetic Relations of Western Nevada Ores. *Trans.* (1905) **36,** 372; *Ore Deposits* (A. I. M. E., 1913), 590.

[57] Blamey Stevens: The Laws of Igneous Emanation Pressure. *Trans.* (1912) **43,** 167; Physical Data of Igneous Emanation, *Trans.* (1912) **43,** 184; and Intrusive Pressure of Mineralizing Solutions, *Min. Mag.* (1914) **11,** 313.

Graton[58] had previously advanced, for a special case, the theory of intrusive pressure in the formation of a deep-seated quartz lens. He was supported in his assumption by Lindgren,[59] who however did not advocate the general applicability of the theory.

Experiment Needed

It is a great pity that the important experimental work of Barus on aqueous silica solutions has not been continued. A fraction of the time expended in making detailed descriptions of unimportant ore deposits would if invested in such experiments undoubtedly have brought to light much data of real value to students of this subject. In a closed vessel the mere heating of the volatiles makes sufficient pressure and that pressure can be measured if the vessel is of the proper shape, as the writer showed in one of the papers cited.

[58] L. C. Graton: Reconnaisance of Some Gold and Tin Deposits in the Southern Appalachians. U. S. Geol. Survey *Bull.* 293 (1906), **60.**

[59] Waldemar Lindgren: The Relation of Ore Deposition to Physical Conditions. *Econ. Geol.* (1907) **2,** 107.

Geology and Utilization of Tennessee Phosphate Rock*

By Richard W. Smith,† Nashville, Tenn.

(Birmingham Meeting, October, 1924)

There are three distinct varieties of phosphate rock, in Tennessee, known commercially as: (*a*) the "brown" rock, which is the residual product of the weathering and natural concentration of certain phosphatic Ordovician limestones; (*b*) the "blue" rock, which is an unaltered phosphatic stratum of Mississippian or Devonian age; and (*c*) the "white" rock, which is the result of chemical replacement and deposition.

Before 1894, South Carolina and Florida had supplied most of the phosphate of the United States, South Carolina leading until that date and Florida from then until the present time. In December, 1893, the blue phosphate of Tennessee was discovered almost simultaneously on Swan Creek, in Lewis County, by two independent parties—by one as the result of a careful and systematic search for a workable deposit of the long known kidney phosphate, and by the other through mistaking it for a "bloom" of coal. Leases were taken and, with further prospecting disclosing other deposits in the region, the development progressed as rapidly as could be expected without railroad transportation.

In January, 1896, Judge S. Q. Weatherly, who was interested in the development of the blue phosphate, while riding along the road south of Mt. Pleasant, noticed some slabs of a brown rock in place above the limestone. He took samples, and had them analyzed, thus discovering the high-grade brown phosphate of this district. This discovery was kept a secret until July, 1896, when mining was started at the village of Mt. Pleasant. Development proceeded rapidly, many owners obtaining their capital by the sale of their rock fences, which were largely composed of slabs of high-grade brown phosphate. The development of the brown rock, which is richer and more easily mined than the blue, caused Tennessee to advance from a poor third in production to second, surpassing the output of South Carolina, and being excelled only by Florida.

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* Published with the permission of the Tennessee Geological Survey. This paper is a synopsis of a detailed report in preparation for publication by the Tennessee Geological Survey.

† Assistant geologist of Tennessee Geological Survey.

Phosphate Horizons in Tennessee

In the following geological column of Middle Tennessee, the phosphate horizons are italicized:

Mississippian
- Waverlian
  - Fort Payne chert
  - New Providence shale
  - Ridgetop shale
  - *Maury green shale* (kidney phosphate)

Mississippian or Devonian
  - Chattanooga "black" shale
  - *Hardin sandstone* (blue phosphate)

Silurian
- Niagaran
  - Lobelville or Louisville limestone
  - Waldron shale
  - Laurel limestone
  - Osgood shale
- Medina
  - Brassfield limestone
- Richmond
  - Fernvale formation
  - Arnheim limestone

Ordovician
- Cincinnatian
  - Maysville group
    - *Leipers limestone* (brown phosphate)

Mohawkian
  - Trenton Group
    - Catheys formation
    - *Cannon formation* (brown phosphate)
    - *Bigby limestone* (brown phosphate)
    - *Hermitage formation* (brown phosphate)
  - Black River Group
    - Carters limestone

Chazyan
- Stones River Group
  - Lebanon limestone
  - Ridley limestone
  - Pierce limestone
  - Murfreesboro limestone

At no one place in the Central Basin are all of these formations present. At some places certain formations are absent because that region was above water at the time and erosion was going on instead of deposition, and the time interval is represented by an unconformity; at other places, because the formations missing were once present but were later eroded before the deposition of the later beds. During Silurian time, most of Middle Tennessee was above water, and the Silurian sea extended

only in narrow arms or embayments along the western edge of the Central Basin, so that the Silurian formations are very irregular in thickness and extent.

The work in recent years of Dr. E. O. Ulrich, of the U. S. Geological Survey, and Dr. R. S. Bassler, of the U. S. National Museum, has added greatly to our knowledge of the stratigraphy of the Central Basin of Tennessee.

Distribution of Brown Rock

In general, the brown rock is confined to the western edge of the Central Basin. The center of the basin has no deposits, because the rocks outcropping there are stratigraphically lower than the phosphate horizons. The eastern edge contains no deposits, because the phosphate horizons are absent or, if present, their character has changed so that they are not phosphatic limestones. For example, along the eastern edge of the Central Basin the Bigby formation is absent, and the Cannon formation, which gives rise to phosphate deposits around Franklin and Nashville, is composed almost entirely of "dove" limestones similar to the basal dove member of this formation at Nashville. In considering the distribution by fields or districts, it must be borne in mind that they represent somewhat artificial divisions which, in many cases, connect with one another directly, without complete interruption.

Mt. Pleasant District

The Mt. Pleasant field is the most striking example of the blanket type of deposit. Its superiority is due to the remarkable development of all the factors that are helpful in the formation of a good blanket deposit. The Bigby formation, which is richer in phosphate here than elsewhere in the Basin, outcrops over a large area having the structure of a low dome with its center east of the town of Mt. Pleasant. The rocks dip away from this crest on all sides, providing good underground drainage to three creeks. These creeks, in places, have cut down to the upper Hermitage formation, about 30 ft. below the undulating surface of the land between; and this, with the easily soluble character of a bed in the upper Hermitage, has provided perfect drainage over all parts of the area. This has resulted in a blanket deposit of from 2 to 10 ft. over much of the area, in addition to the rich lump rock found in the cutters.

Since its discovery, several millions of tons of phosphate have been removed from this district. The end of the industry has often been predicted, and would have been accomplished by now had not improved methods of recovery enabled the companies to rework the entire area, formerly hurriedly gone over for the best lump rock. At present, five large companies are mining and operating washing plants in the Mt. Pleasant district. Their holdings are estimated to be a 20 years' supply.

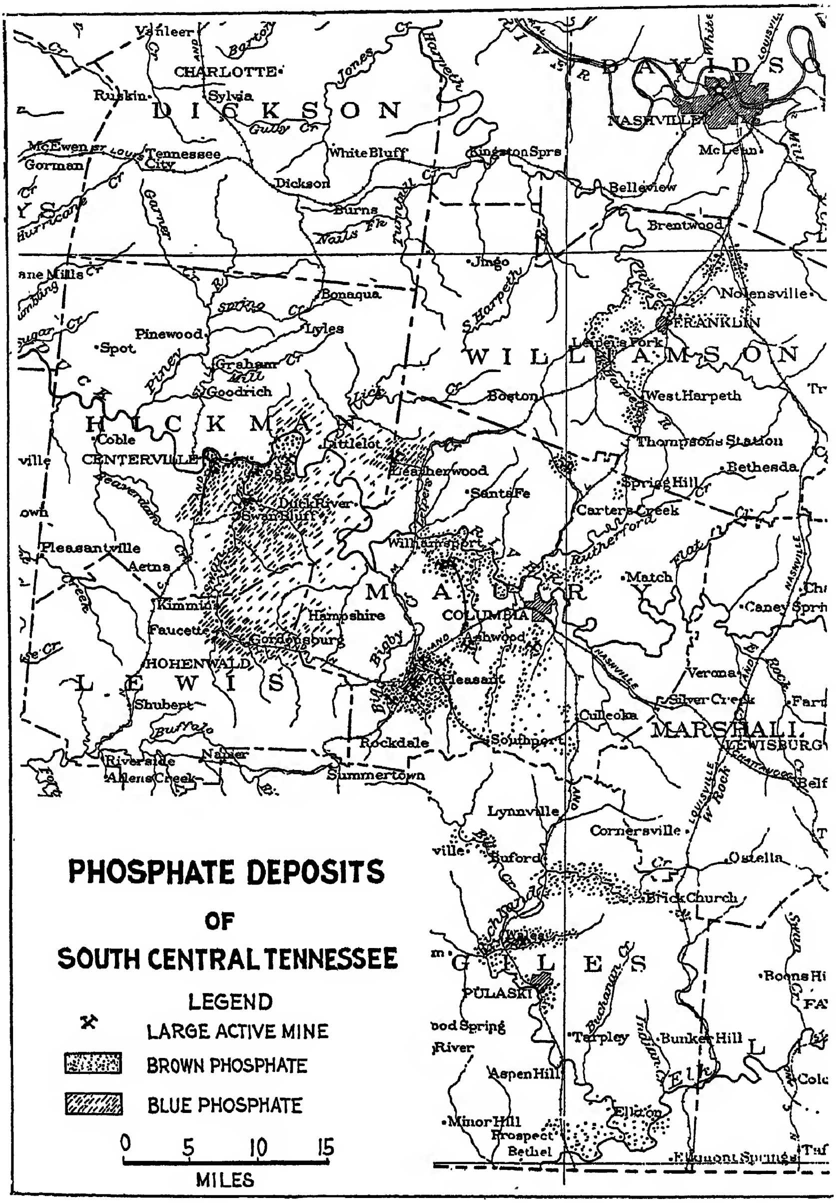

FIG. 1.—PHOSPHATE DEPOSITS OF SOUTH CENTRAL TENNESSEE.

Scott's Mill and Southport Districts

Southeast from Mt. Pleasant, towards Scott's Mill and Southport, the country becomes more hilly, and the Bigby limestone outcrops well up on the sides of the hills, forming numerous rim deposits. These deposits contain quite a tonnage of good grade phosphate but are not easily worked with machinery. On some of them, lump rock is being mined by hand and carted to Scott's Mill, the end of a branch railroad from Mt. Pleasant, where the phosphate is dried on wood and loaded for shipment. Deposits of this character continue east as far as Culleoka on the L. & N. railroad between Columbia and Pulaski.

Wales District

Along Pigeon Roost and Richland creeks, near Wales Station, Giles County, 5 miles north of Pulaski, are numerous deposits of phosphate derived from the Bigby and Cannon formations, some of which approach the blanket type. These deposits are mostly of a sandy and clayey muck containing very little lump rock, and require a careful washing process to keep the grade of the product above 72 per cent. bone phosphate of lime. Two companies have washers here, only one of which is operating at the present time.

Brick Church District

Near the upper end of Richland Creek between Buford Station and Brick Church, several miles northeast of Wales, are extensive deposits of a low-grade siliceous phosphate in the upper Hermitage, Bigby, and Cannon formations. At present these are too low grade for profitable development, but they may be of considerable value in the future. At other places in Giles County are smaller and scattered deposits of similar character that have not been developed.

Columbia District

In the vicinity of Columbia, northeast of Mt. Pleasant, are several deposits of varying size and grade derived from the Bigby formation. On the Williamsport Pike, 8 miles west of Columbia, is a large blanket deposit resembling the Mt. Pleasant deposits. Here two companies have washers, only one of which is operating at present. In its mines the phosphatic stratum is thick but quite deeply buried, and is being mined by the largest drag line in the field, having a boom 125 ft. long.

To the northeast of Columbia, in the vicinity of Bear Creek, are deposits of limited extent of both the "rim" and "blanket" types from the Bigby and Cannon formations. These have been mined to a small extent in the past, but are not being developed at present.

Spring Hill District

West of Spring Hill is a fairly extensive blanket deposit of Bigby phosphate from which the lump rock was mined a number of years ago. Recently a modern washer has been built here and the deposit is being reworked.

Franklin and Nashville Districts

In the vicinity of Franklin and Nashville are numerous small and irregular deposits derived from the Cannon formation and to a less extent from the Bigby formation. From some of these, a little rock has been mined at intervals in thepast.

Centerville and Swan Creek Districts

Along Swan Creek, from Gordonsburg to Centerville, and at a number of places in the vicinity of Centerville, the Leipers formation has given rise to very productive deposits of brown phosphate, largely of the sandy type. At present two companies are operating mines at Twomey, 1 mile south of Centerville.

Other Scattered Localities

At other places, such as Marshall, Lincoln, and Sumner counties, small deposits of poor quality have been found. At Gallatin, Sumner County, some mining has been done in a blanket deposit of Leipers age, but the mines have been idle for a number of years.

Distribution of Blue Rock

The high-grade blue rock lies mainly in Hickman, Lewis and Maury Counties, notably at Gordonsburg on Swan Creek in Lewis County, at Centerville in Hickman County, and at Leatherwood, in Maury County. At present, the mines at Gordonsburg are the only ones operating.

Origin of Brown Phosphate

Igneous rocks contain several phosphate minerals of which the only important one is "apatite," $Ca_5(PO_4)_3(F, Cl)$. It is principally found as a vein mineral or as a product of magmatic differentiation, but is also widely distributed as an accessory mineral. By the action of carbonated waters, the apatite is slowly decomposed and its products carried to the sea, where the phosphatic ingredients are probably soon absorbed into the shells, bones, and tissues of marine animals. When the animals die their remains sink to the bottom, forming slightly phosphatic deposits. The organic remains vary widely as regards their richness in phosphate. Bones and some crustacean remains are very rich, while mollusks and

corals are the poorest, as they consist mainly of calcium carbonate; however, certain brachiopods are highly phosphatic.

The Paleozoic seas of Middle Tennessee contained an abundant invertebrate marine life of which some forms secreted phosphatic shells. Certain of the Ordovician limestones were deposited, according to Hayes and Ulrich,[1] in a very shallow sea, the bottom of which was more or less affected by wave action and tidal currents, and without the addition of much detrital matter from land erosion. In this shallow sea, the deposits were almost wholly of organic origin, consisting in part of the phosphatic shells of small mollusks that flourished abundantly at this time and in part of the more common calcium carbonate shells that form ordinary limestone. Some part of the calcium carbonate was probably redissolved by the sea water, while the less soluble phosphatic shells were rolled and broken by the wave action and tidal currents, and finally deposited on the sea bottom together with more or less carbonate. The rolled fragments of phosphatic shells were probably enlarged somewhat by coatings of phosphate, derived by precipitation from the water which, in turn, had received it from the decomposing animal remains.

Such an origin for the phosphatic Ordovician limestones of Middle Tennessee is clearly shown by their character. They are cross-bedded, due to the current and wave action on the loose material on the bottom of the shallow sea; and are made up of narrow, irregular, alternating bands of relatively pure crystalline limestone, and darker granular, phosphatic layers, both of which contain abundant small fragments of shells. When such a limestone is subjected to weathering by acidulated ground waters, charged with carbon dioxide from the atmosphere and from decaying vegetable matter, as well as the humic acids of the soil, the result is that the calcium carbonate is carried away in solution, leaving behind the more insoluble calcium phosphate as well as the impurities. The extent to which such a weathering takes place depends on the following factors:

1. Position of the phosphatic limestone strata relative to surface topography;
2. Abundant jointing of the rocks, affording channels for the rapid circulation of underground waters;
3. Favorable surface drainage conditions;
4. Character of the underlying and overlying rocks;
5. Presence of clay seams in the limestone.

The position of the phosphatic limestone relative to the surface topography is of utmost importance. If the limestone is buried so deeply by the overlying rocks that weathering cannot reach it, there is no further concentration of phosphate. If the phosphatic limestone lies in a bed too far from the tops of the hills to be completely exposed to surface

[1] C. W. Hayes and E. O. Ulrich: U. S. Geol. Surv., *Geol. Atlas,* Columbia Folio (1903) **95,** 5.

weathering, but only outcrops and is weathered around the sides of the hills, there is formed on this outcrop a deposit encircling the hills, but not reaching to the top. This is called a "rim" or "hat band" deposit. But if erosion has removed the overlying rocks so that the phosphatic limestone itself forms the surface rock, weathering can extend deeper and the areal distribution is not confined to a limited banded outcrop, but is determined more by other conditions. Such deposits are usually more extensive and are known as "blanket" deposits.

Another important factor in the formation of the brown-phosphate deposits is the abundant jointing of the limestones, caused by fracturing as they were elevated to their present position. The underground water, seeping into these cracks and joints, gradually enlarges them by solution of the calcium carbonate of the limestone until they become, in many instances, large underground water channels carrying the water for long distances to the creeks. In this way the so-called cutters are formed. These are the long narrow trenches in the limestone filled with the residual brown phosphate. If the phosphate left is platy, due to alternating bands of varying phosphatic content in the original limestone, the original structure is preserved in the phosphate filling the cutters, the layers sagging in the center because of the removal of the calcium-carbonate portion of the phosphatic limestone. In the Mt. Pleasant field, these cutters are pronounced in their development. In almost any of the larger pits where the phosphate has been mined, there may be seen the long, narrow trenches, separated by the intervening walls of fresh limestone which are irregularly traversed by smaller cross cutters. The main cutters all trend N. 60° W., following the direction of the main joint system of the Central Basin. This is true even with a rim deposit completely encircling a hill, producing parallel cutters on all sides of the hill instead of radial drainage as one might expect.

To produce the deep weathering necessary to form a good phosphate deposit of the blanket type, the main creeks should all be lower than the surrounding country, so that there may be a rapid movement of the underground water through the phosphatic strata to the streams. There should be numerous tributary streams so that every part of the phosphatic strata is well drained. If the dip of the rocks is toward the streams, the rapidity of the movement of the underground water and of its solution powers is increased.

The effect of the character of the underlying and overlying beds is of considerable importance. In the Mt. Pleasant district, the upper part of the Hermitage formation contains a bed of shelly and easily soluble limestone that frequently weathers cavernous, permitting the surface water to drain down through the cutters in the Bigby formation to it and emerge as springs at the outcrop of this bed along the creeks. In places where this bed is absent or is separated from the Bigby by a

thick bed of the usual argillaceous Hermitage limestone, the drainage conditions are not as ideal for the formation of extensive phosphate deposits. Around Franklin and Nashville, the Bigby formation is overlain by a dense, resistant "dove" limestone, which is the basal member of the Cannon formation. This formation is absent in the Columbia and Mt. Pleasant regions. This dove limestone, acting as a resistant capping, has prevented the exposure of broad areas of the Bigby formation like those in Maury County, and has thus prevented the formation of extensive blanket deposits of Bigby phosphate in these regions.

Clay seams are usually quite impervious to water and tend locally to restrict its downward percolation. Thus, if there is a clay seam in the phosphatic limestone, the limestone just above it is subjected to a more than ordinary supply of acidulated water, causing somewhat more leaching action. In this way caves are formed, extending back into the limestone walls, sometimes connecting two adjacent cutters, and leaving unweathered limestone boulders completely surrounded by phosphate.

Origin of Blue Phosphate

The blue phosphate is an unaltered phosphatic portion of the Hardin sandstone outcropping at the base of the Chattanooga "black" shale at the edge of the Highland Rim on the southwest side of the Central Basin. It varies in thickness from a few inches to 3 or 4 ft.; and in composition from a highly phosphatic, oolitic variety to an ordinary shale or sandstone. Unlike the brown phosphate, it is evidently a sediment that has not been altered since its formation. How, then, did it get its phosphatic content? The best clue to its origin is its position. It is present only where the underlying Ordovician limestone (Leipers) is phosphatic. Furthermore, in places where there is an intervening layer of Silurian limestone and the blue rock does not rest immediately on the Ordovician, it is much less phosphatic. It was therefore suggested by Hayes and Ulrich,[2] that just before or during Devonian time there was an interval of erosion, during which brown phosphate deposits, similar to those in the Central Basin today, were formed by the leaching of the phosphatic Leipers limestone. These brown phosphate deposits, on submergence in the sea, would furnish an abundant source of material for the making up of the new beds; and if so situated as to receive no additional sediments would tend to produce a high-grade phosphate rock. Above the Chattanooga "black" shale, the thin Maury green shale contains rounded phosphatic nodules or "kidneys;" these, however, are not of economic importance.

Origin of White Phosphate

The white phosphate is found in Perry and Decatur Counties near the Tennessee River, and in Johnson County in the northeastern part of

[2] *Loc. cit.*, 6.

the state, in formations not present in the Central Basin and therefore not shown in the geological column. In both cases, it is the result of chemical replacement or secondary deposition by waters bearing phosphate derived from slightly phosphatic overlying formations. In the Tennessee River section, it occurs in formations of the upper Silurian and Devonian age; and in Johnson County, in formations of Cambrian age. Owing to its irregular character and limited extent, the white phosphate has never been developed.

FIG. 2.—MINING DEEP CUTTER, HOOVER & MASON MINE, MT. PLEASANT, TENN.

MINING AND TREATMENT OF BROWN-ROCK PHOSPHATE

Brown-rock deposits lie near the surface and are easily worked in open pits. The overburden of soil, clay, and slightly phosphatic muck varies in thickness from almost nothing to such a thickness that its removal is unprofitable. It is generally considered that the economic limit is 6 ft. of overburden for every foot of phosphate found; but of course this depends on the selling price of phosphate, the cost of labor, and the mining methods used.

Early Methods

In the early days of the industry, mining was done entirely by hand. The overburden was removed by hand or by horse scrapers. The phosphate was loosened with a pick and the lumps were shoveled into carts with a "spall fork;" the rich phosphate sand or "muck" and the smaller lumps that would pass through the tines of the fork were allowed

to go to waste. Where the deposit occurred on a slope, a face was usually started on the lower end and worked toward the upper; the overburden, being undermined, was allowed to fall in on the worked ground, and shoveled back.

Before shipping, the rock was dried by piling up on wood, which was then ignited; or by spreading it on the ground in the sun for several days and turning it repeatedly with a plow. With the coming of the larger companies, these crude and wasteful methods were gradually replaced

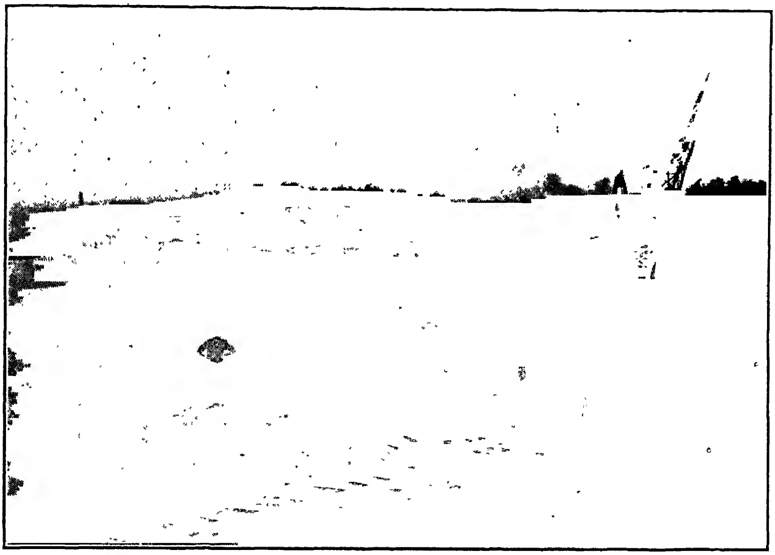

FIG. 3.—MINE OF ARMOUR FERTILIZER CO., CENTURY DISTRICT, ALONG COLUMBIA-WILLIAMSPORT PIKE.

by modern machinery; but they are still used to some extent by individuals mining small and isolated tracts for the lump rock only.

Modern Mining Methods

The larger companies now strip the overburden by drag-line excavators using a 1½ to 2½ cu. yd. Page scraper bucket on a 50- to 85-ft. boom. The overburden is stripped from a long area, the width of the reach of the machine, and dumped on previously mined-over ground at one side.

The entire thickness of the phosphate horizon, lump rock, muck and all, is mined. Where it is thick enough, drag-line excavators are used for the mining; but where the phosphate is thin or occurs in the narrow "cutters" or underground drainage channels between "horses" of

limestone, it is mined by pick and shovel with negro labor. One phosphate company at Mt. Pleasant has designed a machine, called the "cantilever," resembling a traveling crane, for raising hand-filled buckets from deep or narrow cutters and dumping them into the tram cars.

Steam shovels have been tried for both mining and removing the overburden, but because of their limited reach, in comparison, with the drag-line excavators which mine to a depth of 40 to 50 ft., their use has been abandoned.

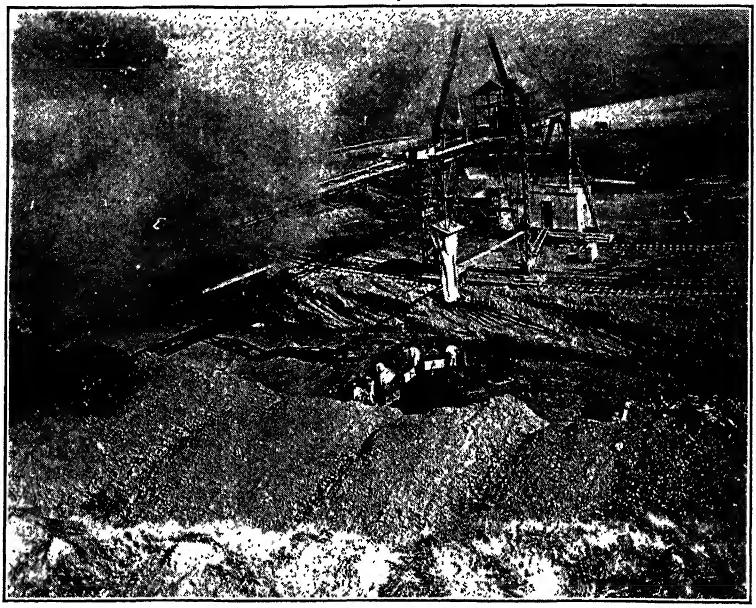

FIG. 4.—MINING WITH CANTILEVER, HOOVER & MASON MINE; OVERBURDEN IS PILED IN MINED-OUT AREA.

The material is transported to the washers by tram haulage, using 10- to 18-ton dinky locomotives on a 36-in. gage line. Either 3-yd. end-dump or 4-yd. side dump cars are used.

At one mine, where the topographic conditions are favorable, the overburden is stripped and the phosphate is mined by hydraulic methods similar to those of the Florida phosphate field. Two streams of water are used with 10-in. nozzles with 1-in. openings. The water is under 120 lb. pressure. After the overburden has been washed into the previously mined-over territory, the phosphate is washed to a depression, or "sump," where it is picked up by a hydraulic jet and carried to a centrally located pump. Here the larger lumps are caught on a 4-in. grid, and the material is pumped to a point from which it can flow to the washer

in a trough. With this method, some trouble has been experienced by flint and iron nodules in the overburden settling down on top of the phosphate horizon instead of flowing off with the overburden as it was removed.

Washing Methods

The muck, or unconsolidated portion of the phosphatic stratum, is a mixture of phosphatic sand and clay, and has a phosphate content of from 30 per cent. to 60 per cent. bone phosphate of lime. As the fertilizer manufacturers demand phosphate running 72 per cent. bone phosphate of lime or over, and with impurities of iron and alumina less than 6 per cent., this material cannot be utilized without washing out the clay and thus concentrating the product until it meets the requirements.

The first attempt at washing brown phosphate was made by the old Tennessee Phosphate Co., at Mt. Pleasant, in 1899 or 1900. This was simply a log washer for cleaning the lumps of their adhering clay. No attempt was made to save the fine material, with the result that the old mud ponds of the washers of this type contained material running from 40 per cent. to 60 per cent. bone phosphate of lime. Most of these have since been reworked at a profit. In 1907 the Independant Phosphate Co. made the first attempt to save the fine material by running the wash water from the log washer through long troughs that settled the coarsest of the phosphate sand. When the plant was shut down, the sand was shoveled out of the troughs. From this time on, considerable attention was given to saving the fine material and improvements were gradually made, until today the washing plants are large and complicated, and the recovery is relatively perfect.

Though the details of manipulation vary at the different plants, the principles of recovery are the same at all. Briefly, it consists of a thorough mixing of the material with water and agitation until the clay lumps are thoroughly disintegrated and the clay and fine sand are in suspension and separated from the coarse sand and lumps. This is followed by a separation of the lump material, the coarse sand, and the fine sand; and a separation of the sands from the clay by gravity with the addition of fresh water at frequent intervals.

The material is usually carried to a crusher at the top of the washer, which partially breaks the large lumps of clay and phosphatic muck. From the crusher the material goes through a mixing process consisting of either double iron log washers or rotating cylinders with flights or shelves that lift the material and drop it back into the water. The object is to break up the mud balls and agitate the material until the clay and fine sand are in suspension and separated from the coarse sand and lumps.

The overflow goes to the fine-sand recovery; the discharge to some form of classifier, usually a revolving screen, which separates the lump rock from the coarse sand. The oversize from this screen runs over a picking belt, where the remaining mud balls and piece of chert and limestone are removed by hand, and the resulting lump rock goes to the wet storage

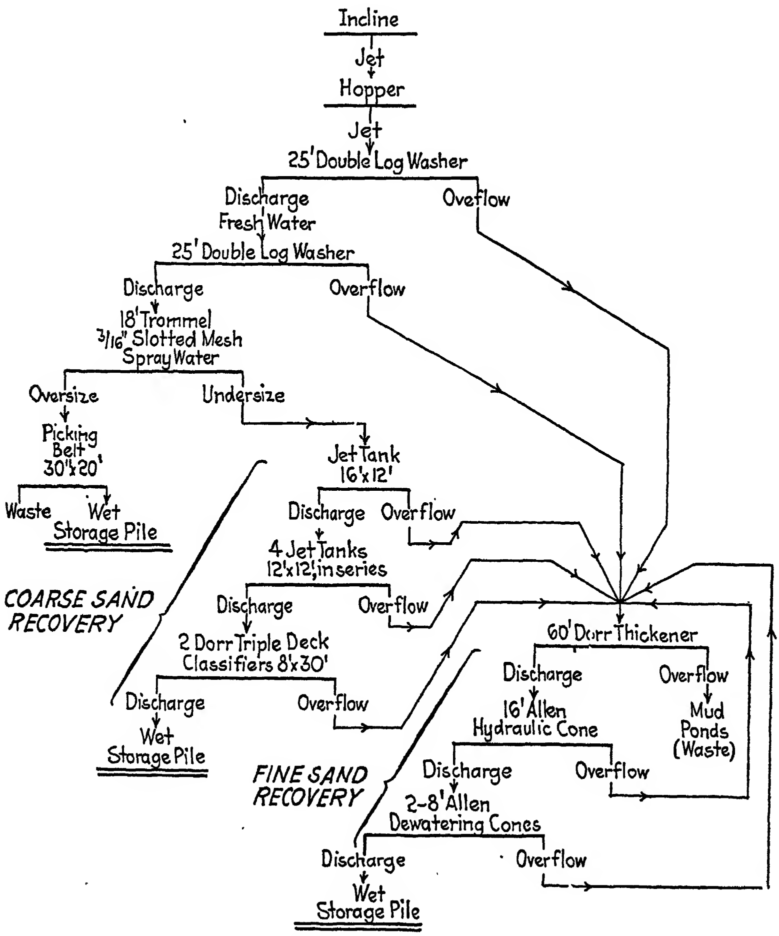

FIG. 5.—PROPOSED FLOW SHEET OF REMODELED WASHER OF INTERNATIONAL AGRICULTURAL CORPN. PLANT AT MT. PLEASANT, TENN.

pile before drying. The undersize from the screen goes to the coarse-sand recovery.

The coarse-sand recovery equipment consists of a series of jet cones, inverted pyramids, cylindrical jet tanks, Allen hydraulic cones, or a combination of two or more of these. The jet cone is an inverted metal

cone with an overflow around the rim, by which the clay and the finest sand are carried away to the fine-sand recovery. The heavier sand settles to the bottom, where a jet of fresh water carries it up through a pipe to the next cone and finally to some dewatering device. The inverted pyramid is the same except in shape. The jet tank is a cylindrical tank, but works on the same principle because the sand settles in the box until the free space for the water is in the shape of an inverted cone.

The last step in the coarse-sand recovery is the dewatering of the washed product by an Allen dewatering cone, a Dorr triple-deck classifier, or a plain drag classifier. The drag classifier is simply an endless drag conveyor removing the settled phosphate sand from a long settling box with an inclined bottom up which the conveyor moves. Whichever device is used, the dewatered phosphate sand goes to the wet storage pile for further draining before drying, while the overflow goes to the fine-sand recovery.

The equipment for the fine-sand recovery consists of a large Dorr thickener, more Allen hydraulic cones, or both. The dewatering of the fine sand is accomplished by more Allen dewatering cones, or a Dorr bowl classifier. This is a combination of a single-deck classifier with a small bowl thickener, placed over the lower end of the classifier tank. With either method of dewatering, the discharge goes to the wet storage pile, while the overflow is carried either to the settling pond or back to the fine-sand recovery.

After draining on the wet storage pile, the washed material is carried, by cranes, to the feed bins of the driers. The driers are rotary cylinders 5 to 7 ft. in diameter and 30 to 50 ft. in length, heated by coal or oil burners at one end. The heat and gases pass through the driers to a stack at the other end; the rock is fed in at either the hot or cold end. From the driers, a conveyor belt or a small tram or trolley car carries the rock to the top of the dry storage shed. Underneath this shed is a concrete tunnel containing a conveyor belt on to which the rock can be fed by gravity. At the end of this conveyor belt, automatic loaders fill the freight cars for shipment. Such is the general process of washing the phosphate. The details vary, and the flow sheet of one company may be more elaborate than that of another. Each company has a different problem to deal with, according to the character of the deposit it is mining.

Cost of Production

The cost of production varies considerably among the different companies and fluctuates from time to time. James A. Barr, engineer for the International Agricultural Corpn., has furnished the following estimate for the general cost of production in the Mt. Pleasant field in 1923:

	Cost per Long Ton
Stripping overburden	$0.50
Mining	0.75
Transportation to washer	0.20
Washing	0.75
Drying	0.75
Shipping and track expense	0.10
Working cost	$3.05
Capital reserve for rock depletion	0.75
Insurance, overhead, depreciation of equipment, etc	1.00
Total cost per long ton	$4.80

If the rock is not owned outright a royalty of $0.50 to $1.00 per long ton must be substituted for reserve for rock depletion.

Method of Prospecting

Because of the irregular character of the brown phosphate deposits, careful prospecting is necessary before development. The area to be prospected is laid off in 50-ft. or 100-ft. squares, and at the intersection of each square a hole is put down to the limestone by means of a post-hole auger. By examining the material brought up, the thickness of the overburden and of the phosphate is determined. The sample of the phosphate thus obtained is washed in as many changes of water as seems necessary to approximate the conditions of a phosphate washer and, after drying, is analyzed. At certain determined places in the phosphate area thus blocked out, pits are sunk to the bottom of the phosphate. From each pit, a sample 1 ft. square and the thickness of the entire phosphate layer is taken. This is washed in tubs, approximating as nearly as possible the conditions of a phosphate washer, and the resulting washed phosphate is weighed. This weight divided by the thickness will give the recovery per cubic foot. The average recovery per cubic foot times the area, in square feet, underlain by commercial phosphate times the average thickness of the phosphate in this area, all divided by 2240 lb., will give the tonnage of phosphate that may be recovered from the area.

Mining and Treating the Blue Rock

The blue rock is workable where a minimum thickness of about 18 in. of high-grade rock is present. Such deposits occur at intervals over the whole area, and are usually connected with one another by the thinner, unworkable portions. Their character as a rock stratum extending into and through the sides of the hills makes it necessary to use underground methods of mining. Main tunnels are driven from the surface into the phosphate and, at regular intervals, rooms about 25 ft. wide

are turned off at right angles, leaving pillars of about the same width. Sometimes double rooms, 50 ft. wide, are worked. After the rooms are completed, the pillars are drawn, allowing the roof of shale to cave. One-ton cars, loaded by hand, are hauled to the mouth of the tunnel by mules, thence by engines, to the crusher.

Where the Chattanooga "black" shale is thin or absent, so that it is necessary to remove the kidney phosphate layer of the Maury green shale when mining the blue rock, it is sometimes possible to make use of these phosphatic nodules. As the blue rock contains little or no clay and chert, it is not washed like the brown, but is simply crushed and dried in the rotary kiln driers.

Uses of Phosphate Rock

Fertilizer

The greater part of the phosphate rock mined in Tennessee is made into soluble "acid phosphate" in various fertilizer factories, located near the market, by treatment with an equal amount of sulfuric acid. In this form, a maximum amount of phosphorus is immediately available as a plant food. It may be used alone or as the basis for complete fertilizers. Most of the larger phosphate-mining companies in Tennessee have their own fertilizer factories scattered over the Southern and Middle Western States.

A recently patented process, called the A. L. Kreiss process, produces a phosphorus-potash fertilizer by fusing phosphate rock with soluble potash salts in a rotary cylindrical kiln at temperatures of 850° to 1300° C. The product is a soluble calcium-potassium phosphate that is said to have the advantage of correcting rather than adding to the acidity of a soil. A plant using this process has been in operation for several years at Lakeland, Florida, making a product that is said to give very satisfactory results to the consumers. It is expected that the process will soon be in operation in Tennessee.

It has been demonstrated[3] that on certain soils phosphate rock, when finely ground and mixed with the soil along with humus in the form of manure or plowed-under crops, becomes available slowly, perhaps through the action of bacteria and the weak acids formed by the decomposition of the organic matter. At points to which freight rates are not too high, the phosphorus is much cheaper in this form, but the result is a gradual building up of the soil rather than a forcing of the immediate crop. To meet this demand, some of the Tennessee phosphate is finely pulverized and sold direct to the farmers, particularly in the Middle Western States, such as Illinois, where the State Agricultural

[3] W. H. Waggaman and C. R. Wagner: Analysis of Experimental Work with Ground Rock Phosphate as a Fertilizer. U. S. Dept. Agr. *Bull.* No. 699 (1918).

Experiment Station has strongly recommended the use of raw ground-rock phosphate.

Raw ground-rock phosphate is finding an increasing use as one of the ingredients of cattle feed, for the purpose of furnishing the necessary phosphorus for bone and tissue building. In 1923, over 20,000 tons of ground phosphate rock was shipped from Tennessee to various feed companies in the United States.

Metallurgical Uses

There has been an increasing demand in the last few years for high-grade lump rock for use by steel plants in the United States to raise the phosphorus content of the steel to the required limits. There has also been an increasing demand for ferrophosphorus, largely used by foundries and basic open-hearth furnaces to obtain the correct phosphorus content in their products. By the use of ferrophosphorus, the phosphorus content of the steel can be easily adjusted without changing the grade of the product. For example, the low-phosphorus steel produced in the basic open hearth is a disadvantage in the manufacture of tinplate, because the steel sticks together when rolled double.

The entire production of ferrophosphorus in the United States is by two furnaces; The Rockdale Furnace at Rockdale, Tenn., 6 miles south of Mt. Pleasant, and the Federal Phosphorus Co. at Anniston, Ala. The Rockdale Furnace is a 55-ft. stack, four-stove blast furnace owned and operated under patents by J. J. Grey, Jr., It is charged with coke, lump phosphate rock, siliceous Tennessee brown iron ore, and some mill cinder and limestone. The fundamental principle involved is that at temperatures of 1300° to 1500° C., with the reducing conditions maintained by carbon or coke, silica assumes the properties of a relatively strong acid in so far as its ability to combine with bases is concerned and, therefore, it can displace the phosphoric acid of phosphate rock forming silicates of lime and free phosphoric anhydride (P_2O_5). By the reduction power of the incandescent coke, elemental phosphorus is produced, which is absorbed by the molten iron. The result is an iron containing from 16 to 22 per cent. phosphorus. It is cast in slabs, instead of pigs, and is broken into convenient lumps. The slag is pulverized by pouring into cold water, and the adhering ferrophosphorus is recovered by magnetic separation.

The Federal Phosphorus Co. obtains the same results by using an electric furnace, thus avoiding infringement of Mr. Grey's patents, which cover the blast-furnace production only. In addition, by maintaining an excess of phosphate rock over that required to saturate the molten iron, it obtains fumes of phosphoric anhydride (P_2O_5), which are collected by a Cottrell precipitator, in the form of liquid phosphoric acid containing 55 per cent. H_3PO_4, for chemical uses.

Chemical Uses

In addition to the phosphoric acid produced at Anniston, a small amount of the best grade of Tennessee phosphate is made into chemically pure phosphoric acid for chemical use.

The Victor Chemical Co. of Nashville, Tenn., is making monocalcium phosphate for use in self-raising flour.

Future of Industry

The phosphate industry has not recovered from the after-war slump. This is largely because of the poor financial condition of the farmers, who since 1920 have bought as little fertilizer as possible. Then, too, the Florida field, having lost most of its export trade, has encroached upon the Tennessee market where the freight rates are favorable. It is hard to estimate when conditions will pick up and production become normal.

The easily mined high-grade deposits of Tennessee are rapidly becoming exhausted. The problem of the future is how to utilize the vast deposits of lower grade material. The Bureau of Soils has done considerable experimenting on the use of an electric furnace or a modified blast furnace to produce phosphoric acid by its volatilization and collection by a Cottrell precipitator. The results of these experiments[4] indicate that such a process will be commercially feasible provided that certain mechanical difficulties can be remedied, such as the chilling of the highly siliceous slag on the hearth.

Such a process will have the following advantages over the sulfuric acid method of producing soluble phosphate:

1. As the presence of silica is necessary in the process, low-grade siliceous phosphates that are unfit for sulfuric-acid treatment can be utilized. There are vast quantities of such siliceous phosphates in Tennessee.

2. Run-of-mine material can be used, dispensing with the costly step of washing the rock, which entails the loss of so much phosphate.

3. The furnace process calls for no sulfuric acid, which under present conditions is hauled to the fertilizer plants as acid and hauled away again as gypsum in acid phosphate.

4. By using the furnace process near the phosphate mines, it is possible to produce a relatively concentrated product that can stand heavy handling charges and the cost of long freight hauls.

There has recently been considerable discussion of the possibility of manufacturing nitrate fertilizer on a large scale at Muscle Shoals. If this is done, the most logical way of marketing it would be in the form of complete fertilizers of the ordinary type, or superstrength if the market can be educated to use them. In Tennessee, within a radius of 30 to 70

[4] W. H. Waggaman: Investigations of the Manufacture of Phosphoric Acid by the Volatilization Process. U. S. Dept. Agr. *Bull.* No. 1179 (1923).

miles of Muscle Shoals, are vast deposits of low-grade siliceous phosphate suitable for the furnace process just described. An electric-furnace plant using electric power from Muscle Shoals could be located at the mines and the liquid phosphoric acid used to treat ground phosphate rock to make triple superphosphate, to which the nitrates could be added in making nitrogenous fertilizers.

The increasing demand for high-grade lump rock for the steel trade and the production of ferrophosphorus will soon be hard to meet. The fine material cannot be used because it sifts down through the charge in the furnace. At present, a number of the phosphate companies are experimenting on processes for briquetting the fine material to supply this demand.

Notes on the Geology of East Tintic

By G. W. Crane,* Salt Lake City, Utah

(Salt Lake City Meeting, September, 1925)

When ore was discovered on the Tintic Standard property in the spring of 1916, new developments were immediately started both north and south of that property, on the supposition that in East Tintic the ore would be found to occur in northerly trending channels, similar to those so characteristic of the older camp. But 9 years of developments have failed to establish the existence of such a channel. This is due undoubtedly to important differences in the geology of the two camps, that were not recognized, and is indicative of the danger in the general tendency toward interpreting the geology of East Tintic too strictly in terms of that of the older camp. It should not be forgotten that while the two sections have had practically the same geological history, the net results of a long series of geologic processes have developed in them conditions that differ so widely and in so many particulars that the necessity of regarding them as essentially different districts seems apparent. In an effort to emphasize this fact, this paper is devoted chiefly to a comparison of the geological conditions as found in the two camps and is addressed more particularly to those familiar with the geology of the Tintic district. Following a short description of the several ore discoveries and of the Tintic Standard mine is a brief discussion of the evidence of ore channels in East Tintic. For my information regarding the Tintic Standard mine, I am indebted to the courtesy and generous assistance of Messrs. Wade and Snow of the Tintic Standard Mining Co.

East Tintic District

The Term East Tintic district is here used to represent that eastern part of the Tintic mining district lying between the productive portion of the old camp of Tintic on the west, the foot hills of Goshen Slope, on the east, the Denver and Rio Grande Railroad on the north, and Silver Pass on the south, all in Utah County. As represented by the geological map, Fig. 1, it includes an area about 4 miles wide, east and west, and 5 miles long, north and south, within which the Tintic Standard mine is about centrally located. The map is a compilation of data supplied, for the most part, by the Chief Consolidated Mining Co. and represents the

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* Mining geologist.

progress of geological mapping to date. Some of the contacts marking subdivisions in the igneous rocks are adopted from the U. S. Geological Survey district map. Geological subdivisions and nomenclature con-

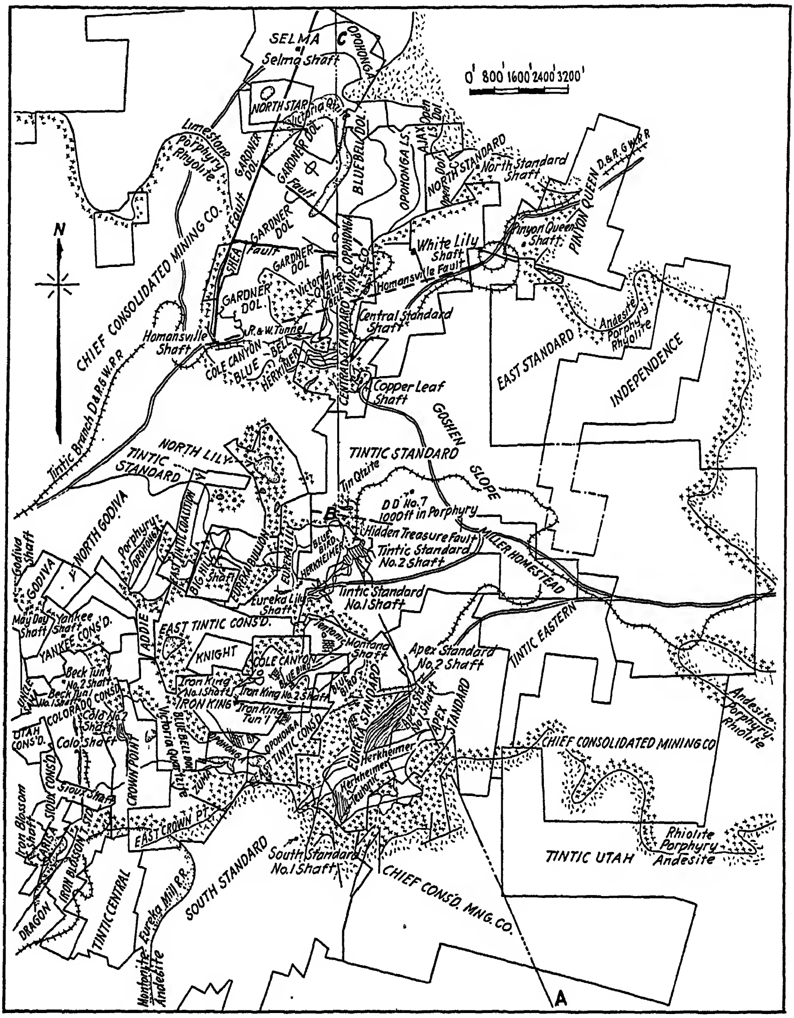

Fig. 1.

form to those used by the U. S. Geological Survey. The property boundaries represent the latest information available.

Surface Formations

One of the most striking, though minor, differences between the camps of Tintic and East Tintic is the character of the surface formations.

In the old camp, the greater part of the surface is composed of limestones, which are bordered on the south by the monzonite stock and on the northeast by overlapping rhyolite-porphyry flows. The deep shafts are sunk from the surface in limestone, except those on the Chief Consolidated property, and the Chief is the only mine in which the orebodies have been traced far beneath the rhyolite capping. In East Tintic, practically three-fourths of the surface rocks consist of rhyolite flows, andesite-latite tuffs and breccias and monzonite porphyry dikes. The remaining one-fourth consists of limestone, shale, and some quartzite in more or less isolated areas or as a part of a narrow, irregular band of sediments reaching from the old camp to the vicinity of the Tintic Standard and Apex Standard mines. As a result, most of the mine workings are either under a capping of rhyolite or more or less associated with intrusive rocks.

Structure

Structurally, also, the two districts differ widely for, while the old camp is situated on the upturned edges of the west limb of a great synclinal fold in which the attitude of the bedding is for the most part vertical and fairly uniform in strike, East Tintic lies on the relatively flat beds of an anticlinal fold in which, as the result of block faulting, the beds strike and drip at all points of the compass. In the vicinity of the Tintic Standard and Apex Standard shafts, the prevailing dips are to the east and at angles up to 45°, indicating that here apparently is the east dipping limb of an anticlinal fold, the axis of which must lie to the west of these properties. In the western portion of the district, the prevailing dips are to the west and here conditions resemble more nearly those in the old camp.

Thickness of Limestones

There are also great differences in the relative thickness of the limestone formations. In Tintic, the limestones extend from the surface to depths of from 4000 to 5000 ft., the deepest workings having still 1000 or more feet to go to reach the Tintic quartzite. In East Tintic, from a large part of the camp the limestone formations have been entirely eroded and are relatively thin over the remainder, being represented in full succession only on the western extremity of the district. In most cases, mine workings south of the Homansville fault have entered the Tintic quartzite formation at depths of from 500 to 1000 ft.

Ore Horizons

Due to different stages of erosion, the orebodies in the two camps occur at widely separated geological horizons. While those in Tintic are found, mostly, in the upper half of the series, the known orebodies of

East Tintic occur at much lower horizons, the limestone beds of the Ophir formation next above the Tintic quartzite being the most prolific. At the Iron King and Apex Standard mines, all the ore mined has come from fissures in the quartzite.

Intrusive Rocks

Perhaps the greatest contrast in the geology of the two camps lies in the relative amounts of intrusive igneous rocks. In the old camp, intrusive rocks, other than the great monzonite stock at its southern end, are found only at depth in the deeper mines, the most conspicuous being the porphyry dikes of the southern end of the lower levels of the Centennial Eureka mine. Except a few small dikes within the monzonite area on the south of the town of Mammoth, no dikes of monzonite porphyry are reported. In East Tintic, on the other hand, dikes of monzonite porphyry are common and cut both the rhyolite flows and the underlying sediments. A series of monzonite outcrops along a line extending from a point about ½ mile south of the Zuma mine, northeasterly through the Zuma, the Iron King, the North Lily, and the Central Standard Consolidated mines properties to a point on the Denver & Rio Grande Railroad near Canyon Siding mark a zone of monzonite porphyry intrusion approximately 3½ miles in length which, in its bearing on the origin of the orebodies of East Tintic, is easily the major geological feature of the camp. While the outcrops of the monzonite are generally small, the mine workings show the dikes to be much larger at depth, constituting in some places the bulk of the country rock. This is true of the 1200-ft. level of the Zuma mine and of the west portion of the 1600-ft. level of the Iron King mine. In the Copperleaf workings, several monzonite porphyry dikes were encountered at depth though none are exposed at the surface. This is indicative of the great strength of the zone and of its probable extension to considerable distance beyond the range of surface exposures. At the Zuma mine, the limestone-monzonite contacts carry fair sized stopes of silver-lead ore, definitely establishing the genetic relation of the ore-depositing solutions to the dikes and strongly indicating that this zone of monzonite intrusion is to East Tintic what the monzonite stock to the south of the town of Mammoth is to the older portion of the district.

Faulting

All sections of the Tintic district, where the sedimentary rocks are exposed, exhibit a great deal of faulting, most of which took place during the period of folding and prior to the period of mineral deposition. East Tintic, however, exhibits in addition certain well-defined post-mineral faults which, in the Tintic Standard mine, displace the orebodies for vertical distances of 10 to 80 ft. These, in connection with several

post-rhyolite faults that have been mapped in the area 1 mile to the south of the Apex Standard mine, indicate that East Tintic has suffered considerable deformation of a recent type which, superimposed upon the earlier faulting, adds greatly to the complexity of local structural problems. Post-mineral faulting is only of doubtful occurrence in the old camp nor has any faulting of the rhyolite contacts been observed there.

Occurrence of the Ore

In the mode of occurrence of the ore as known to date, conditions in the two camps are quite different. Aside from the fissure veins in the monzonite to the south of the town of Mammoth, the known orebodies of Tintic occur in the form of channels in the limestones, which in their grouping constitute four or more great ore zones. There is not an orebody in the camp that is not readily assignable to one or another of these zones. In East Tintic, the ore discoveries are apparently isolated occurrences, each having its own peculiarities as to horizon and structural relations and in their distribution do not fall into lines of mineralization such as characterize Tintic ore channels. On the contrary, they are more particularly of the fissure-vein type, deposited apparently from mineralizing solutions of a local source.

Ground-water Levels

Ground-water level conditions are quite unlike in the two sections of the district. In the deep limestone mines in the older camp, groundwater is usually encountered about 4800 ft. above sea level and all mine workings below this level are drained by pumping. In East Tintic, there appears to be no definite ground-water level and no water at all in those mines bottoming in the Tintic quartzite. This is accounted for by the differences in the structure of the two sections and in the permeability of the formations involved. The uniform water level of the old camp is due apparently to the joint influence of the synclinal structure and several shaly limestone horizons, which together form an impervious basin. No such water basin exists in East Tintic and the absence of water in the Tintic quartzite horizon, even at levels 400 ft. or more below the usual water-table and 100 ft. below the level of Utah Lake, indicates that the quartzite formation is probably dry to great depths. In view of this condition, it is not unlikely that the water level of the synclinal trough of the old camp is superficial only and that should mining operations penetrate to the quartzite no water would be found there.

Mine Temperatures and Strata Gas

Wherever the mine workings of the East Tintic district have entered the Tintic quartzite formation, there have been found relatively high-

rock temperatures and considerable strata gas calling for special equipment for the ventilation of the mines.

In the old camp, rock temperatures in the limestone increase only at the normal rate of about 1° F. for each 100 ft. in depth and rarely is there any gas. Only when driving extra long drifts into new territory is induced ventilation necessary.

Oxidation Products

The high-rock temperatures and strata gas of East Tintic are attributed, in part, to the oxidation in place of the sulfide ores and, in part, to the oxidation of the sulfides of iron in the Ophir shale without the presence of water. In the old camp, oxidation in the presence of more or less ground-water has taken the normal course, forming carbonates and hydrous oxides with the liberation of but little heat and practically no byproduct gases. In East Tintic, oxidation in the absence of water has been restricted largely to the formation of sulfides and byproducts of carbon dioxide and nitrogen gas, with the liberation of much heat. The shattered quartzite forms a natural reservoir for the gases, which, being heavier than air, sink into recesses of the country rock. But with the approach of a storm and lower barometric pressures, the gas expands and, pouring from its rocky retreat, rapidly floods the lower levels of the mines and would soon prevent all operations were it not for vigorous ventilation. As the barometric pressure rises, the gas retreats to lower zones and, with continued rise, reaches its former level. In time, the ventilation process depletes the supply of gas to a point where it gives but little trouble, indicating that the volume of gas originally encountered was the accumulation of ages and that additions from present-day oxidation are very slow.

Pebble Dikes

Another feature in the geology of East Tintic and in which it differs from the old camp, is the so-called "Pebble Dikes;" a phenomenon which has excited no little interest as to their origin and relation to the ore deposits. In its simplest form, a pebble dike is a vein, or dike-like ledge, of well-rounded pebbles of quartzite, shale, limestone and porphyry fairly well cemented in a matrix of arcose sands of about the same materials. In East Tintic, these dikes have a nearly north-south trend and, ranging in width from a few inches to 4 or 5 ft., cut both the sedimentary formations and the overlying porphyry flows, frequently extending to depths of 1200 ft. or more. Developments at the Homansville shaft, however, showed that many dikes exposed at the surface were not found when undercut at a depth of 600 ft. The dikes are particularly a feature of the central portion of East Tintic, being most abundant in the vicinity of the Tintic Standard and the Copperleaf mines. As

one proceeds to the higher slopes to the west, occurrences are progressively less frequent. At the Tintic Standard mine, a 4-ft. pebble dike cuts vertically through a large orebody now being mined on about the 900-ft. level and where in contact with the ore shows no evidence of having been attacked by the ore-depositing solutions. At the Tip Top Manganese mine, Homansville Canyon, a small, well defined, pebble dike cuts both the porphyry and the underlying limestones and splits the orebody without evidence of having been mineralized. In both cases it seems apparent that the dikes are younger than the associated mineralization and, therefore, have no bearing on either the origin or the localization of the ore deposits. Out of the evidence at hand, I am led to believe that they were probably formed by shore-line gravels filling fissures in the country rock due to adjustments to early periods of faulting along the Wasatch Range and that they afford additional evidence in support of Gilbert's theory of the existence of lakes at higher levels and still more ancient than Lake Bonneville.

Developments

At no time in its history have development opportunities in East Tintic been more in demand. While the Tintic Standard mine still stands alone as the one steady producer in the camp, several discoveries have been made recently and during the last six months initial shipments have come from the Iron King, the Apex Standard, and the Zuma mines. With active developments in progress at a number of properties, including the Eureka Standard, the Copperleaf, the North Standard, the Crown Point and the East Yankee, additional discoveries may be looked for in the near future.

Apex Standard Mine

Developments at the Apex Standard mine consist of two shafts and and about 7000 ft. of laterals. No. 1 shaft is 900 ft. deep and No. 2 shaft 1500 ft. to the north of it is 950 ft. deep. They are connected by a drift from the 900-ft. level of the No. 1 shaft, which affords good ventilation. All the work at No. 1 shaft is on the 900-ft. level or therefrom. At No. 2 shaft, laterals have been driven from the 800-ft., 875-ft., and 950-ft. levels. Both shafts pass through porphyry and Ophir shale and bottom in the Tintic quartzite formation. Most of the workings are in the quartzite and shale, but one drift driven 600 ft. east of the No. 1 shaft enters the Teutonic limestone formation.

Where not ventilated, the mine is hot, normal rock temperature being about 120° F.; during periods of low barometric pressure strata gas tends to obstruct operations on the lower levels.

Between the two shafts, there are three northeast-southwest faults, two of which dip to the north. Ore has been found between the 800

and 900-ft. levels in the vicinity of both shafts, impregnating the fractured planes of the major faults near the contact of the Ophir shale and the Tintic quartzite formations. Shipments during January and February, 1925, total about 136 tons of silver-lead ore of an average grade of 1.25 oz. gold, 14.2 oz. silver, and 19.3 per cent. lead with gross values of $44.34 per ton. The gangue minerals are quartz and barite and the ore is for the most part thoroughly oxidized. In Fig. 2, the Apex Standard workings are shown in vertical section.

Zuma Mine

Developments at the Zuma mine consist of a 1300-ft. shaft and about 8000 ft. of drifts and raises, with laterals from the shaft at the 200-, 500-, 1200-, and 1300-ft. levels. Much work has been done on the 1200-ft. and 800-ft. levels, the latter being reached by a 610-ft. incline raise from the 1200-ft. level and a 250-ft. incline from the 500-ft. level.

The country rock is Opohonga limestone, very white and thoroughly recrystallized as the result of the intrusion of numerous monzonite-porphyry dikes. Geological conditions are about the same on all levels, except with depth the porphyry dikes are wider and make up about half of the rock exposures. Not having entered the Tintic quartzite the mine has normal rock temperatures and very little, if any, strata gas.

Ore was first encountered on the 500-ft. level and has been traced downward to about the 1050-ft. level, the best stopes lying between the 700-ft. and 800-ft. levels. It occurs in pipe-like shoots on the limestone-monzonite contacts, apparently replacing the limestone, and rakes downward to the southwest on a pitch of about 50°. Ore is now being mined from the 700-ft. level and on the 1200-ft. level a drift is being driven southeasterly in an attempt to intercept the ore at that depth.

The ore contains gold, silver, lead, and some copper; the gangue minerals are quartz and barite, the former predominating. It is only slightly oxidized. Shipments to June 15, 1925, total 113 tons of dry ore. The last shipment of 40 tons gave returns of 0.016 oz. gold, 19.95 oz. silver, 3.5 per cent. lead, and 0.71 per cent. copper per ton.

Iron King Mine

From time to time, there have been shipped from the Iron King mine for fluxing purposes considerable iron and manganese ore, which was obtained from the porphyry-limestone contact in the vicinity of the old shaft. But the ore in which we are primarily interested is the gold-silver type recently discovered in the vicinity of the new shaft. Here the ore impregnates a wide north-south zone of highly fractured Tintic quartzite and cements the plane of a strong N 20° W fissure within the fractured zone. Ore has been mined from the fissure by an incline raise from the 1565-ft. level to the 1300-ft. level and work is being done through a con-

nection on the 1060-ft. level of the No. 2 shaft. Recently, a drift from the 900-ft. level picked up the same mineralized fissure in the limestone members of the Ophir shale formation; here the ore carries some lead. Shipments to date total about 360 tons of dry ore with an average gross value of $12 per ton; in this ore, the gold content was characteristically high.

Eureka Lily Mine

Prior to 1913, the Eureka Lily mine produced over 50 carloads of lead ore and about 1000 tons of zinc ore. The lead ore averaged 42 per cent. lead and from 2 to 4 oz. of silver to the ton.

The mine is opened by a 500-ft. vertical shaft with laterals at the 70-, 130-, 330-, and 500-ft. levels and an incline winze from the 500-ft. level with laterals at the 1400- and 1600-ft. levels. The shaft is sunk inDagmar and Teutonic limestones and the ore was mined from between the 70- and the 330-ft. levels, occurring as a replacement of the limestone along a north-south fissure in the Teutonic formation. Loughlin,[1] describes the ore on the 70-ft. level as occurring in bunches and boulders up to 1 ton in weight in a gangue of decomposed limestone of a yellowish brown character but which contained no quartz. On the 330-ft. level the ore, he states, consisted of galena and zinc blend and contained 10 to 15 oz. of silver to the ton. The gangue was chiefly quartz with some barite and fragments of unreplaced wall rock.

The incline winze is sunk on a north-south mineralized fissure that dips about 45° to the west and carries, at intervals, small bunches of ore. At the 1400-ft. level, it passes from the Ophir shale into the Tintic quartzite, the beds here dipping about 15° to the east. Developments on the vein on the 1400-ft. level disclosed a seam of copper-silver-lead sulfides from 2 to 12 in. in width and crosscuts in the foot and hanging walls of the vein revealed several parallel mineralized fissures at intervals of about 50 ft., the intervening quartzite being highly fractured and impregnated with pyrite. Developments on the 1600-ft. level showed conditions similar to those on the upper level and work was stopped without further discovery. The mine has not been operated during the last five years.

Tintic Standard Mine

The mine is opened by two shafts. No. 1, an old prospect shaft sunk 1000 ft., is now used for ventilation only. The No. 2, the main working shaft, is 1441 ft. deep and a winze sunk 159 ft. below the bottom level gives the mine a total vertical depth of 1600 ft. The collar of the No. 2 shaft has an altitude of 5996 ft.; the bottom of the winze, therefore, lies at an altitude of 4396 ft., or about 100 ft. lower than the surface of Utah Lake. However, there is no water in the mine. The main levels are

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[1] Bureau of Mines *Prof. Paper* 107.

from 113 to 132 ft. apart and the No. 2 shaft is about centrally located with respect to the orebodies so far developed. The total length of the mine workings exceeds 7 miles.

Ore was discovered in the spring of 1916, since which time the mine has been a steady producer. The total production from its inception to Dec. 31, 1924, was 18,368.901 oz. of gold, 16,666,540.34 oz. of silver, 139,552,143.59 lb. of lead, and 4,425,569.75 lb. of copper. The total dividends paid to July 1, 1925, amounted to $4,829,870.25, of which $1,267,166 was paid in the first six months of 1925. Shipments for the year 1924 totaled 156,397.42 tons, valued at $6,721,575.15. Operating profits were $3,170,242 and net income to surplus was $2,252,322.74.

Character of Ore

The ore contains chiefly silver and lead with some gold and copper. That which is too low grade for direct smelting is treated at the company's mill at Harold, Utah, 12 miles east of the mine. The milling process is chloridizing roast and leaching in modified form after the Holt-Dern process. The direct-smelting product includes lead ore and dry ore. The average grade of the three classes of ore shipped during the 12 months of 1924 was as follows:

	Gold, Ounces	Silver, Ounces	Lead, Per Cent.	Copper, Per Cent.
Lead ore	0.037	30.35	28.87	0.336
Dry ore	0.094	97.96	3.31	0.62
Mill ore	0.032	18.61	5.22	0:37

Geology

The surface formations in the immediate vicinity of the Tintic Standard mine include outcrops of Herkimer and Dagmar limestone with overlapping Packard rhyolite flows on the east slope of the hills. Along the line of the rhyolite-limestone contact, there is considerable iron-stained jasper and silicified limestone, with here and there a small deposit of iron and manganese oxides forming, as it were, a promising surface showing. This mineralization, though generally interpreted as earlier than the period of ore deposition, I understand was what led to the location of the ground and its untiring exploration.

A section of the No. 1 shaft shows Herkimer, Dagmar, and Teutonic limestones to where it crosses the South fault at a depth of 665 ft., thence to the 1000-ft. level it is in Tintic quartzite. The No. 2 shaft is sunk in rhyolite to a depth of 594 ft., thence in Teutonic limestone and Ophir shale to a depth of 1400 ft., thence to the bottom in Tintic quartzite. The incline winze sunk from the 1450-ft. to the 1600-ft. level at a point 185 ft. southeast of the No. 2 shaft is all in Ophir shale. About 90 per cent. of the mine workings from the No. 2 shaft are in the Ophir formation.

Structure

In no mine in the Tintic district has faulting been a greater factor in the localization of the orebodies than in the Tintic Standard. The major faults are conveniently grouped into three systems; an east-west, a northeast, and a northwest. The east-west and the northeast systems are undoubtedly premineral, while the northwest is believed to be post-mineral and has every appearance of having faulted the ore.

The east-west system includes two major faults, known as the north fault and the south fault. The former lies north of the No. 2 shaft, for the most part, but strikes north of east and easterly and dips about 45° to the south, cutting the shaft at the 1400-ft. level. The throw is down on the south, the vertical component being at least 800 ft. The south fault cuts the No. 1 shaft 35 ft. above the 700-ft. level, but lies wholly south of the No. 2 shaft. It strikes N 65° E to N 80° E and dips normally about 45° to the north. Below the 1250-ft. level, the dip is much steeper, approaching 90° on the 1450-ft. level. The throw is down on the north, the vertical component being about 900 ft. Neither the north nor the south fault faults the rhyolite. Most of the mine workings lie within the segment between these faults, the relation of which is shown on the N 22° W section through the No. 2 shaft, Fig. 2. A third fault belonging to this series crosses the Hidden Treasure claim about 1500 ft. to the north of the No. 2 shaft, but no underground workings have yet encountered it.

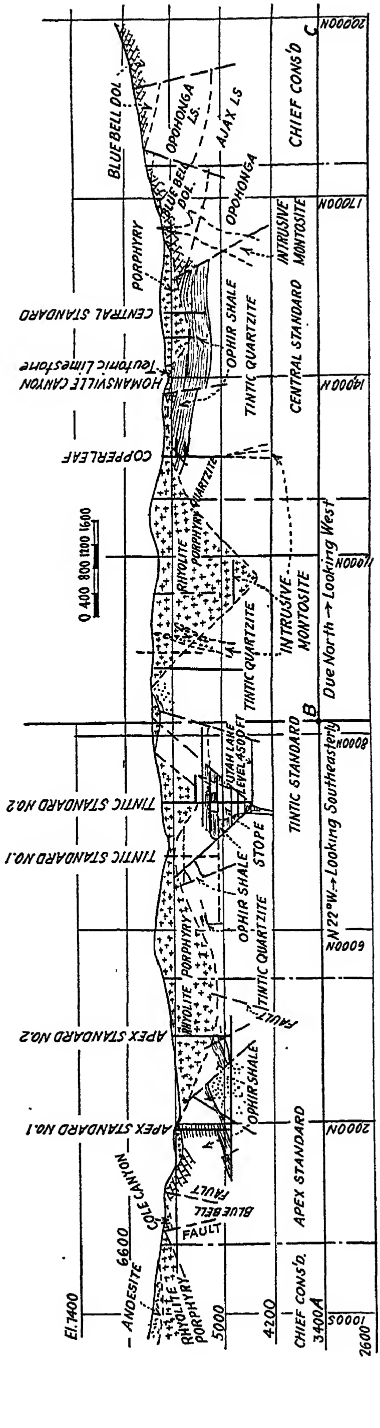

Fig. 2.

The northeast system of faulting includes a series of three or more fissures, which strike from N 25° E to N 40° E and dip 60° or more to the northwest. These fissures are best developed on the 800-ft. level and in the country east of the shaft where the orebodies lie in contact with the quartzite. They are thought to be younger than the east-west system, but premineral in age; because they form the locus of the high-grade orebodies, they are presumed to have formed the main channels for the mineralizing solutions.

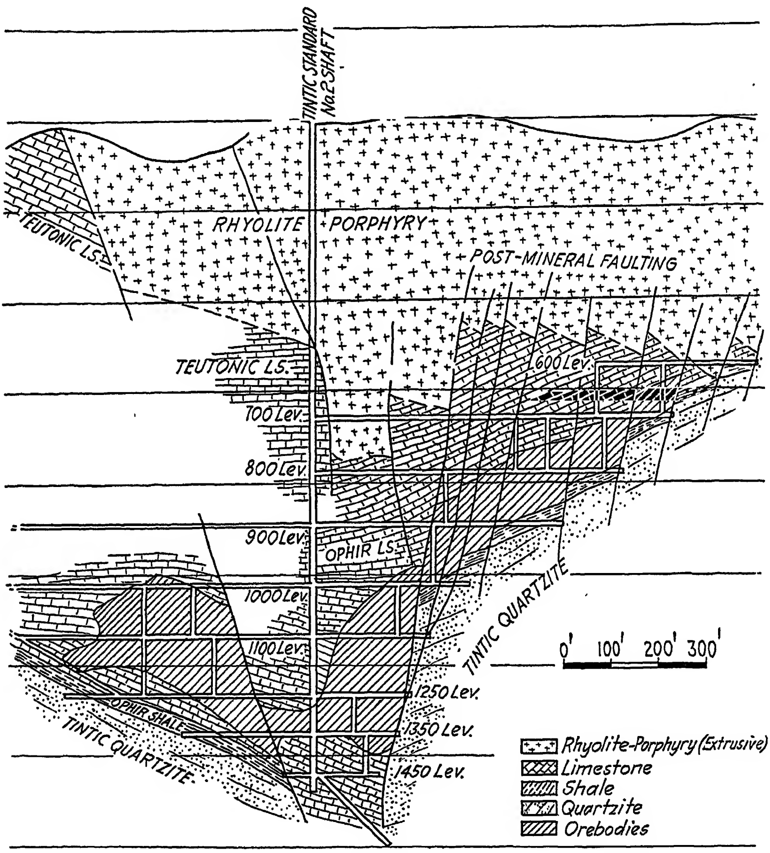

FIG. 3.

The northwest system includes seven or more faults with strikes varying from N 20° W to N 35° W and dips of 80° or more to the southwest. The displacement is down on the southwest with throws of 10 to 80 ft. Faults of this type have been encountered in that part of the mine east of the No. 2 shaft and are especially marked above the 1100-ft. level. Their effect is to step the orebodies progressively up to the east, as shown by the northeast section, Fig. 3. The distribution of the ore segments,

the character of the fault planes, and their sharp demarkation of ore and mineralization limits, together with the offsetting of the older northeast system of mineralizing fissures, mark them as almost certainly postmineral in age. In that case, they should be found to fault the overlying rhyolite.

The combined effect of the three systems of faulting has been to depress an inverted elliptical cone of limestone and shale into a basin formed by walls of quartzite. The No. 2 shaft is almost centrally located with respect to the cone and the drag dip of the bedding is generally toward its center, forming a synclinal basin. In fact, a vertical section drawn in any direction through No. 2 shaft should reveal about the same type of structure. The lowest point of the cone is in the vicinity of the bottom of the 1450-ft. incline winze, which reaches the 1600-ft. level. Below that, structural conditions are still unknown.

Orebodies

While the first shipment from the Tintic Standard mine was obtained from a winze sunk on the south fault from the 1000-ft. level of the No. 1 shaft, the real discovery was made in a deeper winze at a point 410 ft. southwest of the present site of the No. 2 shaft. This was on the tenth floor above the 1250-ft. level. Subsequent developments have shown the ore to occur in a series of roughly rectangular blocks which, beginning at the third floor above the 1000-ft. level at a point 400 ft. southwest of the shaft, extend downward to about the sixth floor above the 1450-ft. level in the immediate vicinity of the shaft, thence upward for 800 ft. to the east and northeast by a series of step faults of the northwest system to about the 600-ft. level.

In the early stages of mining operations, attention was directed more particularly to the recovery of the high-grade oreshoots in a much larger body of lower grade material. Single stopes, as developed to date, have floor dimensions of thirty sets in two directions. As stoping is advanced upon the intervening low-grade areas, they are connected and the ultimate result will be one continuous stope extending from southwest to northeast over a horizontal distance of 1200 ft. and through a vertical range of 800 ft.

The ore always occurs close to the Tintic quartzite, presumably replacing the limestone members of the Ophir shale formation, and locally is seen to replace the shaly members of that formation as well. No ore has been found above the confines of the Ophir formation; and while it always lies close to and in some places directly in contact with the quartzite, it does not enter the latter far except on open fissures, and then for a few feet only. There has been no stoping in the quartzite.

The present line of development is directed to tracing the ore farther northeast and upward on the step faults; also to following it to the south-

east and northwest along the quartzite and shale fault contacts; also to prospecting the Ophir formation in the areas to the north of the north fault and to the south of the south fault on the 700-ft. and higher levels.

Evidence of Ore Channels in East Tintic

With the exception of certain iron and manganese deposits of the type mentioned in connection with the Iron King mine, all the known ore deposits of East Tintic are depositions from magmatic solutions. In that respect, as well as in their mineral composition, they are like those of the old camp, but from what has been said regarding their manner of occurrence and their relation to known geological structures, we can hardly assume that they are parts of ore channels. This is particularly the case at the Tintic Standard mine where, because of the peculiarity of its occurrence, its size and the seeming local source of the mineralizing solutions, it is difficult to conceive of the orebody as only a link in an ore channel. On the contrary, the unusual concentration of so large an orebody in a structure apparently ideally suited to restrict deposition suggests not only a local source of mineralizing solutions but that the local receptacle was amply large to meet all deposition requirements. Again, where the ore occurs in the form of fissure veins in the quartzite, it is certainly quite different from the usual Tintic type, but this does not imply that good orebodies may not be found in the quartzite, particularly where it is highly fractured. However, the ore zones, as we know them in Tintic, have been formed by the replacement of limestone and not in the quartzite.

Conditions favorable to the formation of ore channels appear to have been substantial thicknesses of the replaceable limestone formations and their unbroken continuity in the line of the major structures. Over much of the East Tintic district, the combined effect of folding, faulting, and erosion has been to remove most of the limestone and to leave the remainder in more or less isolated blocks in places nearly, or quite, surrounded by quartzite. In the Goshen Slope area, the limestones, we have reasons to believe, have been entirely eroded and the rhyolite capping is not only thick but probably rests directly upon the quartzite. In the extreme eastern part of the district, because of easterly dips in the formation, there may be more limestone but the eastward sloping pre-rhyolite erosion surface promises great depth to the rhyolite capping and occasions much uncertainty as to what lies beneath it. Perhaps the most likely place in the East Tintic district for the occurrence of an ore channel of the Tintic type is in that western portion which lies between the zone of monzonite intrusion and the old Iron Blossom-Colorado channel. Here geological conditions are, for a space, essentially like those in the old camp; and while it has been little prospected, conditions relative to the source of mineralizing solutions appear to be as favorable as for any

section of the Tintic district. But if, as suspected, the source of the mineralizing solutions is from deep within the zone of monzonite intrusives, then new orebodies of the Tintic Standard type may be looked for at almost any point in the areas marginal to that zone wherever conditions as to structure and formation are favorable.

DISCUSSION

G. W. Crane.—The dikes are made up of pebbles of both sedimentary and igneous rocks; generally the former are the more prevalent but, almost invariably both kinds are present. As a rule, the pebbles are smooth, well rounded, and of all kinds and sizes, just as on a beach. Some dikes contain flat pebbles of slaty Ophir shale that, when picked out of the dike, split into thin layers. Some of the pebbles are of limestone of different colors and textures, but most of them are hard glassy quartzite.

It has been suggested that the dikes have an origin akin to the igneous activity in the district and are related to the phenomenon of ore deposition. This would infer that they were formed from angular rock fragments derived from depth and were rounded by abrasion resulting from the agitation set up by gases escaping through open fissures. If that were the case, there would have been considerable classification or sorting of both the materials of abrasion and the pebbles themselves. On the contrary, the dikes consist of pebbles ranging in size from that of a pea to 4 or 5 in. in diameter, which as a rule, are quite uniformly packed in a matrix of fine sands and coarse gravels which may or may not be cemented. In fact, they have the usual appearance of open fissures filled with beach pebbles.

The dikes are confined largely to East Tintic and are most abundant in the lower eastern portion of the district; farther west, in the higher hills, they practically disappear. They are also more abundant at the surface than at depth. In the Homansville shaft workings, six or eight pebble dikes showing on the surface are represented by only two or three at a depth of 600 ft. at which depth also they are smaller.

H. E. Havenor, Salt Lake City, Utah.—Some years ago, when following one of these dikes I found a gradual gradation in the size of the material from the west toward the east; also the matrix became more siliceous and in some places would carry an assay value.

There is unquestionably some relationship between these pebble dikes and the ore deposits of the district. On the 1200-ft. level of the Copperleaf mine, to the west of the shaft, is a rhyolite dike with this so-called pebble dike on both walls. In another place on the same Tintic property, on the surface, one of these pebble dikes extends over 1600 ft.; so does a small seam of rhyolite-porphyry. In view of the fact that there are evidently concentrations of ore in the district and that these dikes are more or less restricted to that area, in all probability there is some genetic

relationship. One man who worked in the Tintic Standard, said he had found evidences of these dikes fingering upwards without an extension to the surface and with an extension downwards; this would be evidence of intrusive origin. On a property in Sonora, which I mapped very carefully, a dike 400 ft. in width is an intrusive rock containing pebbles. I have found pebble dikes not more than 2 in. in diameter fingering through the limestone, but in that case it cannot be intrusive though very similar in many of its characteristics.

G. W. Crane.—If these dikes have an igneous origin, the matrix should be of a rhyolitic intrusive nature, enclosing angular fragments of the country rock and there would be no reason for the rounded pebbles nor for the great variety of both the sedimentary and the igneous pebbles. The deepest occurrence of a pebble dike that has come to my notice is at the Chief mine at a point about 1400 ft. northwest of the No. 2 shaft. There at a depth of 1800 ft. and cutting diagonally across a drift is a dike of rounded pebbles about 18 in. wide and standing nearly vertical; it is fairly uniform in width and carries perfectly rounded pebbles of all kinds including several pebbles of bird's eye porphyry. Because of its great depth and apparent isolation, this occurrence is of unusual interest.

In some places dikes cutting the rhyolite are not silicified in any degree and the pebbles lie loose on the surface; in other cases, they are well cemented and form ledge outcrops. Where in direct contact with the orebodies of the district, they show no sign of mineralization. At one place on the 900-ft. level of the Tintic Standard mine, a 4-ft. pebble dike cutting a large orebody is perfectly barren; in Homansville Canyon, a 10-in. dike cutting directly through the Tip Top manganese deposit is clean cut and unmineralized. In both cases, it is evident that the dikes are younger than the ore deposits; if such is the case they can have little or no bearing on the origin or factors bringing about the localization of the orebodies of the district.

N. O. Lawton, Salt Lake City, Utah.—The dikes found on the 1200-ft. level of the Copperleaf workings are of monzonite; are those at depths of less than 1200 ft. monzonite or rhyolite?

G. W. Crane.—We speak of them as monzonite; they are very similar to the intrusives found in the Iron King mine where we are warranted in calling them monzonite. They are largely porphyritic, but that is apt to be the case in a small dike and thus subject to rapid cooling.

N. O. Lawton.—In the Copperleaf mine, all the pebble dikes can be traced to the surface; I have also seen these pebble dikes in American Fork Canyon.

G. W. Crane.—That is very interesting, because American Fork Canyon would be well within Gilbert's old Lake basin province.

Iron Fields of the Iron Springs and Pinto Mining Districts, Iron County, Utah

By Duncan MacVichie,* Salt Lake City, Utah

(Salt Lake City Meeting, September, 1925)

The iron fields described here are located in the Iron Springs and Pinto mining districts, Iron County, Utah. This region is in southwestern Utah, about 260 miles south from Salt Lake City, and is reached by the Los Angeles & Salt Lake railroad of the Union Pacific system. The iron fields lie at an elevation of 5500 to 7500 ft. above sea level, and are essentially desert in character. They are covered with a light growth of sage brush, piñon pine, and scattered cedar. Weather conditions are unusually favorable for all-year mining operations, the winters being mild with little or no snow; and the summers not excessively hot, the temperature rarely rising to 100° during the day, while the nights are invariably cool.

The iron-ore bearing area extends from the Iron Springs district to the Iron Mountain mining district, being approximately 25 miles long in a northeast-southwest direction, and from 1 to 3 miles wide in a northwest-southeast direction. In the Iron Springs and Pinto mining districts, the ore occurs at various points around three sides of Granite Mountain and Iron Mountain. This is readily understood by reference to Figs. 1–4. These mountains are composed of andesite, which has been forced up through the surrounding sedimentary formation, in the form of laccoliths.

An extensive report on the geology of these districts was made by Leith and Harder,[1] who state:

> The three dominating geological features of the district are three large andesite laccoliths which constitute the Three Peaks, Granite Mountain, and Iron Mountain; lying in a northeast-southwest line across the area in question.
>
> Three unconformable sedimentary series, aggregating 4000 ft. in thickness, outcrop in successive rings around these laccoliths, dipping outward asymmetrically, very steeply at the contact, less steeply farther away.
>
> Still farther from the laccoliths, lava-flows 2000 ft. thick rest in nearly horizontal attitudes on the tilted sedimentary rocks. These general relations are modified by faulting.
>
> All of the rock formations of the district are more or less covered on the middle and lower slopes by unconsolidated and partly consolidated erosion debris, both aqueous and subaerial, which spreads out on the lower ground to make the deserts.

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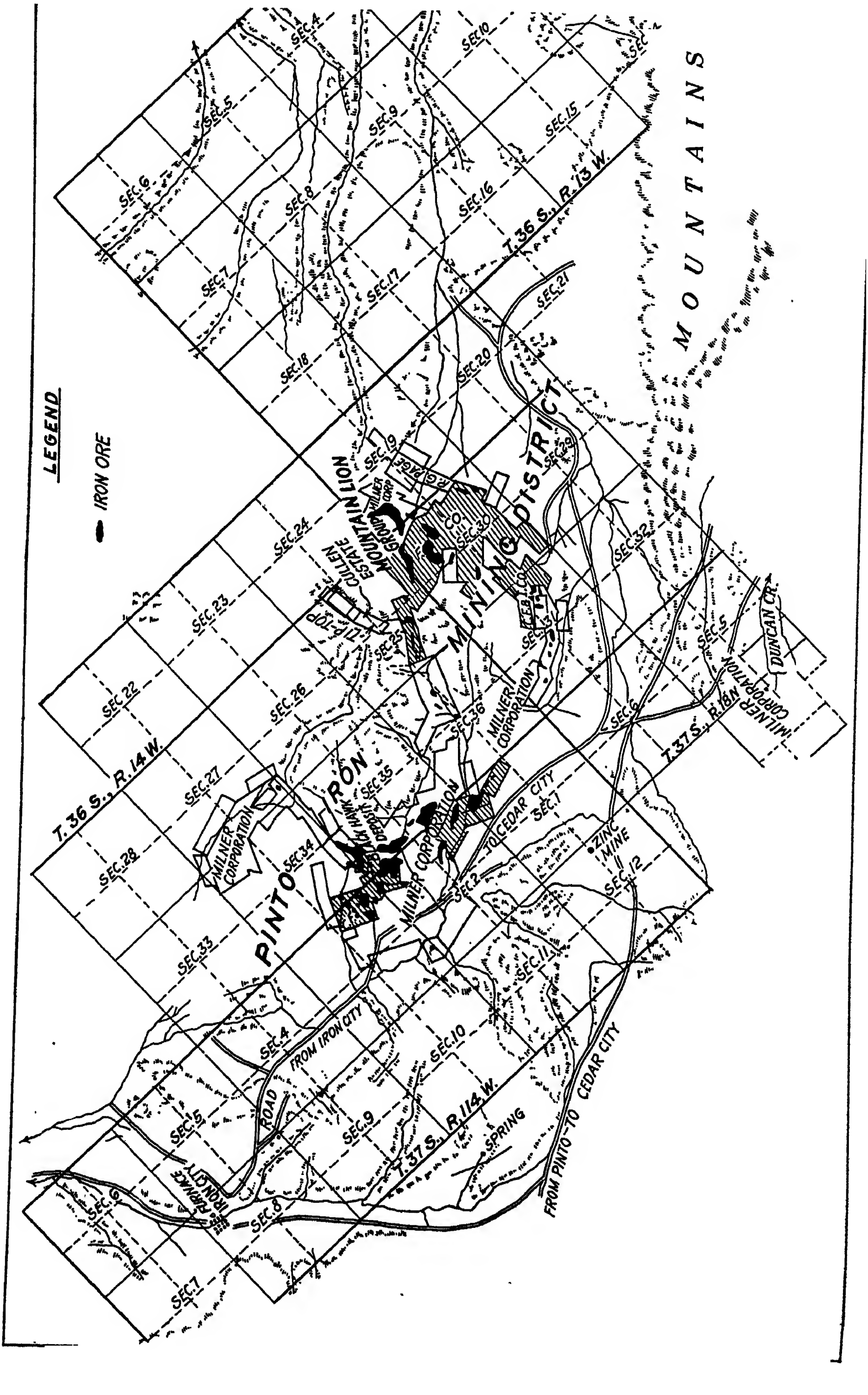
* Mining and consulting engineer.

[1] U. S. Geol. Surv. *Bull.* No. 338.


LEGEND
IRON ORE
MOUNTAINS
PINTO IRON MINING DISTRICT
T. 36 S., R. 13 W.
T. 36 S., R. 14 W.
T. 37 S., R. 13 W.
T. 37 S., R. 14 W.
MILNER CORPORATION
CULLEN ESTATE
LION MOUNTAIN GROUP
BLACK HAWK DEPOSIT
DUNCAN CR.
ZINC MINE
SPRING
TO CEDAR CITY
FROM IRON CITY
ROAD
IRON CITY
FURNACE
FROM PINTO TO CEDAR CITY

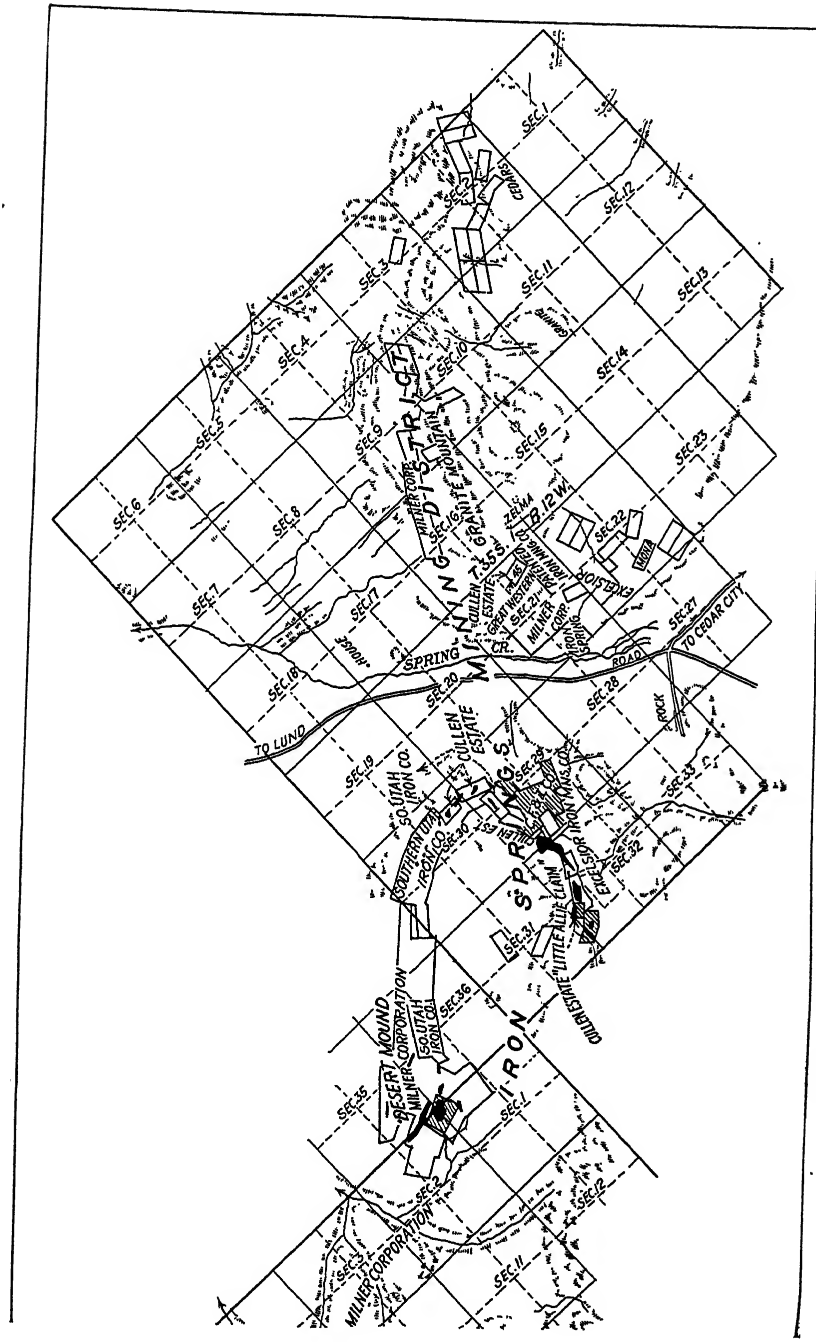


FIG. 1.—IRON SPRINGS AND PINTO IRON MINING DISTRICTS, IRON COUNTY, UTAH.

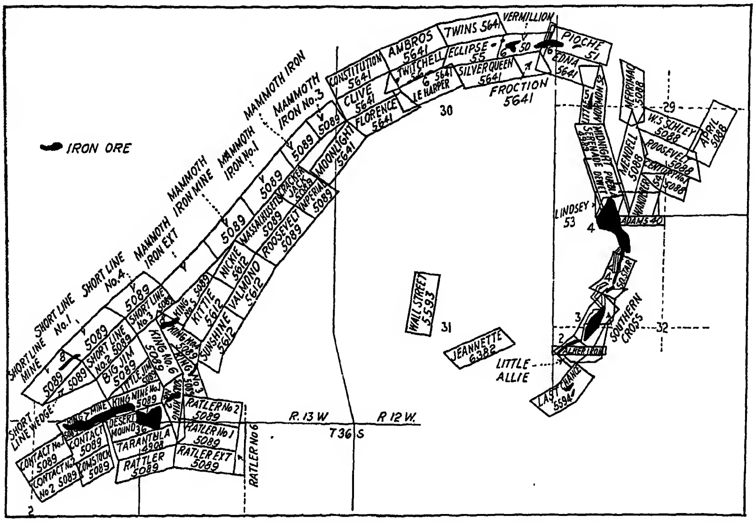


FIG. 2.—LINDSEY HILL TO DESERT MOUND, GRANITE MT., IRON SPRINGS MINING DISTRICT.

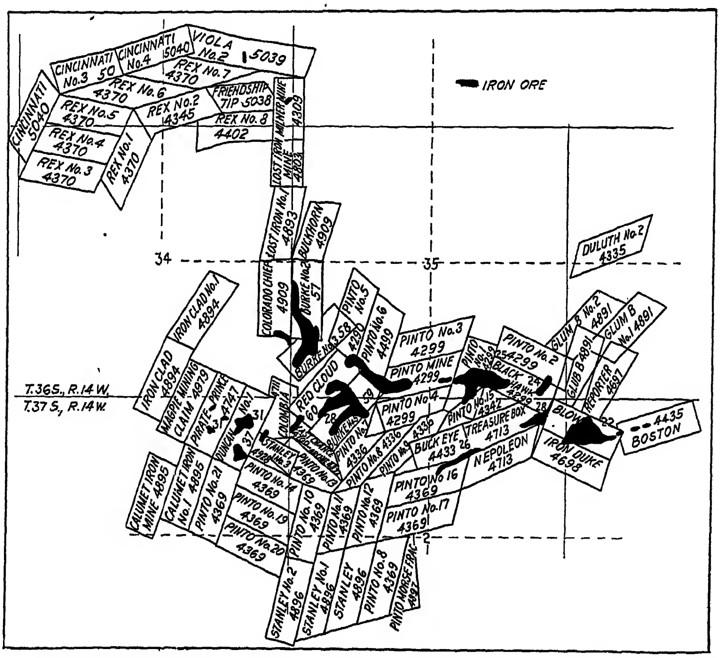


FIG. 3.—SOUTH AND WEST SLOPES OF IRON MT., PINTO IRON MINING DISTRICT.

The laccoliths are exposed only on their upper parts—at no place has erosion shown their bottoms in section. The rock is an andesite of remarkably uniform texture and composition.

Within the area of the laccoliths are a few veins of iron ore and fault-blocks of ore, and Carboniferous and Cretaceous sediments.

Further on they describe the different ore occurrences as follows:

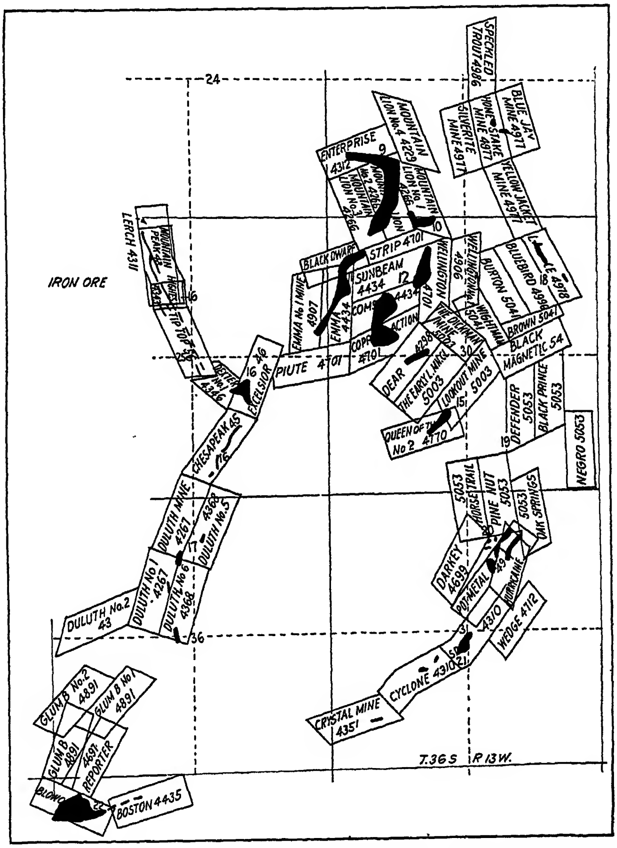


FIG. 4.—EAST SLOPE OF IRON MOUNTAIN, PINTO IRON MINING DISTRICT.

The ore occurrences are found:

1st, in fissure veins in the andesite;

2d, in fissure and replacement deposits along the contact of the andesite with the Carboniferous limestone;

3d, as breccia cement in the Cretaceous fault fissures.

The large deposits are those found along the limestone-andesite contact, with their longer diameters in general paralleling the contact. They are commonly lens-shaped, but there are many irregularities due to faulting and other causes.

In the writer's experience, gained in developing the iron orebodies now being worked by the Columbia Steel Corpn., and from observations in the field, the following conditions would seem to be fairly uniform:

First,—the iron found in the fissures in the andesite is in quantity too small to be considered of economic value, though varying in width from 2 to as much as 12 ft. This ore is practically all magnetite, hard, and difficult to extract. Second,—the iron ore found in the fissures in the lime, and as a replacement of the lime paralleling the contact of the Carboniferous limestone and the andesite, constitutes the deposits of economic importance.

Surface indications would lead one to the conclusion reached by Leith and Harder; that is, that while there were some small bodies of iron ore "separated from the andesite by a layer of the silicated phase of the limestone," the larger bodies of iron ore were lying directly on the andesite. The development work of the Columbia Steel Corpn. has shown, however, that at a short distance below the surface limestone is found abutting directly against the andesite, and this limestone forms the bottom, or foot wall, of the orebodies. The work shows that most of the important iron orebodies are replacements in the limestone and are

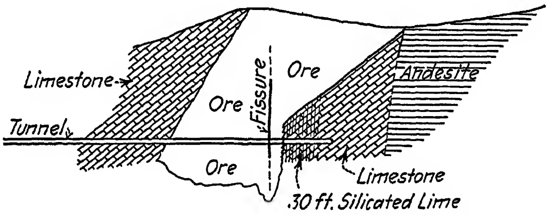


FIG. 5.—IRON SPRINGS DISTRICT, NORTH-SOUTH SECTION, LOOKING EAST.

separated from the limestone-andesite contact by intervening and paralleling belts of silicated lime; and further, that while the andesite at the contact dips at approximately 80°, the limestone and iron orebodies dip outwards at from 30° to 40°, as shown in Fig. 5.

The hanging wall of the iron orebodies is fairly well defined, but on the foot-wall side, in nearly all the developments made by the Columbia Steel Corpn., there is no well-defined line of demarkation. The ore is found to retain its iron content up to a fairly well-defined line along the hanging wall, but on the foot-wall side the iron content is replaced more and more by silica and lime, until the silicated phase gradually merges into the normal limestone resting on the andesite. The intervening belt in which this gradual change of mineral content takes place is a soft clayey material.

There are indications that the 40° dip is not continuous. There are vertical fissures in the ore deposit parallel to the strike, but the limestone on its dip does not cross these fissures, neither does it change its dip. As the ends of the limestone members are exposed, where cut by the fissures, the limestone seems to have been replaced by ore, which follows the fissure on its downward course. This would seem to indicate that at

depth the iron orebodies are on the andesite contact again, but present developments are not sufficient to verify this theory.

Intrusions of andesite extend from the main body of the laccoliths into the foot wall, and quite often reach into the iron orebodies. Where these intrusions occur, the limestone is metamorphosed and glassy and the iron ore is harder and of a higher grade.

Faults are frequent, most of them being at right angles to the strike of the ore deposits, and apparently caused by the cooling of the laccoliths. None of the faults found in the ore deposits up to the present time have been so extensive as to interfere seriously with mining operations.

Origin of the Iron Ores

In Chapter VII of *Bulletin* No. 338, Leith and Harder set forth the apparent origin of the iron ores:

> The principal ore deposits, viz, those near the contact of the andesite and limestone, are partly replacements of limestone. The original bedding of the limestone has been preserved in the ore in a number of places, and there is gradation between the ore and the limestone. These deposits are also in part fillings of fissures in limestone or between limestone and andesite. Where ore occurs within the andesite it fills fissures. The source of the iron-bearing solutions is the same for the limestone replacements and for the vein fillings in the limestone and in the andesite, for their mineralogical and textual characters are the same and in a few cases they are actually connected.
>
> Several hypotheses as to this source have suggested themselves:
>
> 1. That the ore-bearing solutions were associated with the intrusion of the andesite as "igneous after-effects;"
> 2. That they were meteoric waters, cold, or heated by contact with the laccolith, acting after the laccolith intrusions and before the eruption of the surface flows;
> 3. That they were hot solutions, magmatic or meteoric, or both, connected with the late eruptives of the district, deriving the ores from the effusives or from the underlying rocks;
> 4. That they were cold meteoric waters later than the effusives;
> 5. That they were due to some conbination of these sources.
>
> The source of the ore is best explained by the first hypothesis, but later concentrations of the ore have occurred in the order named.

CONCENTRATION AND ALTERATION

Deposition of Ore Following Laccolith Intrusions as Igneous After-effects

Introduction of the Ore

The general association of the ores with the andesite and their specific association with fissures and faults in the andesite and the immediately adjacent limestones, the nature of the ores and gangue materials, especially the primary association of magnetite with garnet, amphibole, pyroxene, mica, apatite, iron sulfide, and glass, seem to allow but one general interpretation, and that is that the ore-bearing solutions were hot, rising from a deep-seated source through fissures in the andesite not filled with ore, at a period closely following the crystallization of at least the outer part of the laccolithic mass.

That the ore was introduced after the hardening and crystallization of the andesite and not before is shown not only by its occurrence in clear-cut fissures in the andesite and by the metamorphism of the andesite near the ore but also by the lack of anything in the nature of a basic edge in the andesite, by the lack of irregularity in its composition, by the absence of ore for considerable intervals along the andesite contact, and by the fact that the intrusion of the andesite metamorphosed the limestone in a clearly recognizable manner, recrystallizing it, decarbonating it, rendering it more siliceous, and indurated the Pinto sandstone to a quartzite spotted by the segregation of ferrous iron in the form of amphibole—all before iron was introduced.

The conclusions of Leith and Harder, as to the origin of the iron ore, are confirmed by the conditions exposed east of the Three Peaks, near the extreme northeast end of the iron fields, where the sedimentaries—together with any large ore deposits that may have existed in the overlying sedimentaries, and even a portion of the andesite laccoliths—have been entirely eroded leaving the hard magnetite iron ore exposed in vertical parallel fissures in the andesite. Here the iron ore has withstood the erosion and in many places is standing as much as 12 ft. above the present surface, the width of the ore varying from 2 to 6 ft. This ore, with the exposed upper portions, apparently continues its downward course in the fissures in the andesite laccolith, and it is a noteworthy fact that practically all of the ore-bearing fissures in the andesite laccoliths are entirely, or nearly, vertical.

Description of Individual Ore Deposits

The ore found in the larger deposits averages in character approximately 60 per cent. hematite and 40 per cent. magnetite, being harder near the surface of the deposit, and softer as depth is obtained. The figures here given are an average of analyses obtained by a sampling of the entire district, by R. W. Dickman of Chicago, Ill., Lerch Brothers of Hibbing, Minn., and C. H. Gibbs, of Salt Lake City, Utah: Iron 58.00 per cent., phosphorus, 00.20 per cent., silica, 5.00 per cent., moisture, 3.00 per cent.

It is the writer's opinion that as work in the district progresses, the economic orebodies will be found to be those that have the large outcrops. These are remarkably large orebodies with little or no overburden. The ore from the Black Hawk deposit is softer than the ore from any of the other bodies, and apparently contains a larger percentage of hematite.

There are twenty-seven separate deposits in this area, but only a few of the larger ones will be considered as there is a similarity in all of them.

Pioche-Vermillion Deposit, Iron Springs District

This is the orebody now being operated by the Columbia Steel Corpn.; it has already been described.

Lindsey Hill Deposit

This deposit, located on Granite Mountain, Iron Springs district, has a surface area of approximately 505,900 sq. ft. It is one of the larger deposits. The iron ore rises on its strike 200 ft. vertically in a horizontal distance of 400 ft. The north side rises at an angle of about 32° and the south side at an angle of about 22°. It has a horizontal width, at its summit, of over 500 ft. with an easterly dip.

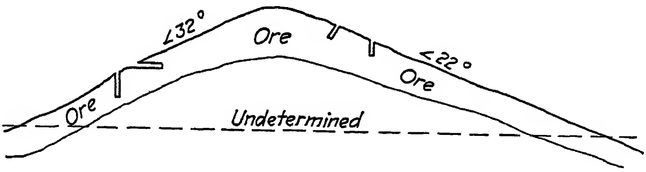


FIG. 6.—IRON SPRINGS DISTRICT, NORTH-SOUTH LONGITUDINAL SECTION THROUGH LINDSEY HILL; RISES 200 FT. IN HORIZONTAL DISTANCE OF 400 FT., DISTANCE ACROSS OREBODY ON TOP OF HILL IS OVER 500 FT.; SECTION LOOKING EAST.

Developments on this deposit consist only on a few short tunnels, shallow pits, and one shaft down to a depth of 44 ft. but these are sufficient to show that there is one intrusion of andesite into the orebody and that a body of lime lies between the andesite and the iron ore. Due to the disintegration of the exposed iron ore and the spreading out of the eroded material on the surface, one would conclude that the iron ore extended to the contact of the andesite, but the development work shows that the lime is between the ore and the contact. A longitudinal section of this deposit is shown in Fig. 6.

Desert Mound Deposit

This deposit, located 3 miles southwest of Iron Springs, has an estimated surface area of 468,000 sq. ft. It is one of the largest surface showings in the entire district. Developments by shafts and shallow pits indicate that bunches, or blocks, of limestone are included in the ore.

Mountain Lion-Enterprise Deposit

This deposit is located approximately 13 miles southwest of Iron Springs; its surface area is approximately 532,000 sq. ft. It is located on the northeast side and near the foot of Iron Mountain, and differs materially from any other of the deposits in that it dips at a much lower angle. The surface of the ore dips at an angle of 17° to the southeast, and the orebody lies in a depression undoubtedly caused by a faulting of the andesite. Where the andesite is exposed adjacent to the surface of the iron deposit, it has the same dip as the iron ore.

Though the sedimentaries are entirely eroded from a large part of the orebody, there is every indication that a large area is still covered by

them. In the southeast portion of this deposit, there is a large body of magnetite which has withstood the erosion, so that it stands approximately 75 ft. higher than the adjacent hematite. It would seem that previous to the erosion of the sedimentaries and the upper portion of the hematite, the surface of the deposit as a whole had an elevation compar-

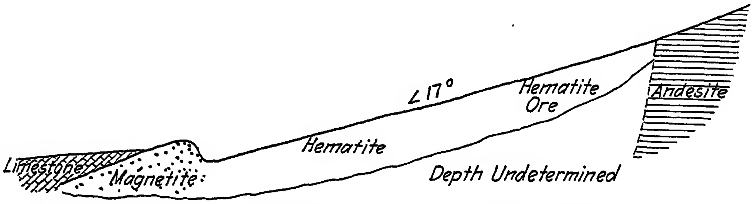


FIG. 7.—MOUNTAIN LION-ENTERPRISE DEPOSIT; NORTH END SLOPE OF IRON MT., 13 MILES SOUTHWEST OF IRON SPRINGS, SECTION LOOKING EAST.

able to the height of the magnetite that has withstood the erosion, as shown in Fig. 7.

To the east of this outcrop of magnetite, the ore dips under the overlying Homestake limestone. There are strong evidences in portions of this deposit to indicate that a belt of lime underlies the ore and rests on the andesite, and also that a portion of the ore of this deposit rests directly on the andesite.

Black Hawk-Pinto Deposit

This deposit, located approximately 16 miles southwest of Iron Springs, and having a surface area of 317,000 sq. ft., lies toward the southwest end of the iron fields. A longitudinal projection is shown in

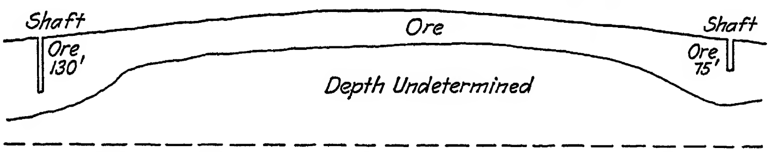


FIG. 8.—BLACK HAWK-PINTO DEPOSIT, EAST-WEST SECTION, LOOKING NORTH.

Fig. 8, and a cross-section in Fig. 9. The projection shows the conditions as indicated by the ores exposed at the present time, but the cross-section is more of an ideal section, as sufficient work has not been done to determine the dip of the ore either at the foot wall or the hanging wall. Neither has the work done determined whether or not there is a belt of limestone between the ore and the andesite. Two shafts have been sunk, approximately 1500 ft. apart, but these do not show the limits of the ore on the strike.

The width of this deposit is given as 480 ft., which represents the width of the deposit on the summit of the hill where the ore is hard, so there is no

question that this width is the minimum estimate. Due to the slope of the surface and the disintegration of the ore, it is difficult to determine the limits of the deposit beyond the figures given, but the indications are that this deposit has a greater width at depth.

The ore of this deposit is softer than the ore from most of the other deposits and undoubtedly carries a larger percentage of hematite. This orebody is easily accessible and is very attractive from an operating standpoint.

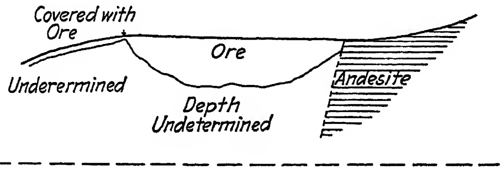


FIG. 9.—BLACK HAWK-PINTO DEPOSIT, NORTH-SOUTH SECTION, LOOKING WEST.

Mining

Mining at Iron Springs, at the present time, is carried on by the open-pit system. A tunnel has been driven under the known orebodies, and raises put up to the surface. The ore is broken around the raises and drawn through chutes at tunnel level into cars of 4-ton capacity. The cars are hauled by electric locomotives to the crushing plant near the portal of the tunnel. From the crushing plant, the ore is transported, on a belt conveyor, to the receiving bins, from where it is discharged into standard-gage railroad cars.

Mining at Desert Mound is carried on with steam shovels, which practice will probably prevail for a long time. The only difficulty encountered so far is due to the numerous faults, joints, and fissures, which cause the ore to break in large blocks, necessitating short holes in stoping, and some blockholing.

Under the present methods, many millions of tons of iron ore can be mined from the Iron Springs and Pinto fields, before underground operations become necessary.

Ore Deposition and Enrichment at the Magma Mine, Superior, Arizona*

(Secondary Enrichment Investigation Publication No. 17)

M. N. Short,† Washington, D. C., and I. A. Ettlinger,‡ New York, N. Y.

(New York Meeting, February, 1926)

The Pioneer mining district, better known as the Superior district, from its principal town, is located in Pinal County in south-central Arizona about 80 miles east of Phoenix and 22 miles west of Miami. The region embraces a confused series of mountain ranges of moderate

Fig. 1.—Superior, looking north from south bank of Queen Creek. Magma mine is indicated by waste dump in gulch to right of center.

height but precipitous slopes, between which lie irregular valleys filled with alluvial detritus. The average elevation is about 2800 ft. in the valleys and the summits of the ranges reach 2000 ft. above this.

Superior is situated on the north bank of Queen Creek where it emerges from one of those ranges into a more or less basin-shaped valley. The drainage of Queen Creek is to the southwest and the flood waters spread out and lose themselves in the desert northeast of Florence, the county seat of Pinal County.

The Magma copper mine (Fig. 1) is the only producer in the district. Although discovered in 1875, its production has been important only

* Published by permission of the Director of the U. S. Geological Survey.

† Assistant geologist, U. S. Geologist Survey.

‡ Mining geologist.

since 1910, when it was acquired by its present owners. The present monthly output (1925) is about 2,250,000 lb. of copper, 50,000 oz. of silver, and 600 oz. of gold.

Previous and Present Investigations

Although the Magma mine is an important producer, surprisingly little has been written about it. The first description was given by F. L. Ransome, who visited the district in 1912.[1] He recognized the main geological features of the district and correlated the formations with those of Globe.[2] At the time of his visit, mining operations had reached a depth of only 800 ft. A brief description of the geology of the locality is given by D. H. McLaughlin, who visited the mine in 1916.[3] This description is not readily available, however, and in 1925 mining explorations have gone 1000 ft. deeper than at the time of McLaughlin's visit. In 1922 the mine was visited by John C. Anderson, who elaborated a theory of primary zoning in the Magma vein but made no attempt to give a description of the general geology of the region.[4] More recently Messrs. Browning and Snow have given a detailed description of the processes of mining and extraction with notes on the geologic relations of the vein.[5]

The authors' interest in the district originates in the fact that both are former employees of the Magma Copper Co. The areal geology of the district was mapped by Mr. Ettlinger as a part of his duties as chief engineer of the company. He was assisted in part in the underground geological mapping and in the preparation of the maps and sections by Mr. Short. The chalcographic work on the ores was done by Mr. Short in connection with a study of the origin of deep-level chalcocite, which was carried on in the laboratory of economic geology of Harvard University and completed in April, 1923. A report on the results was submitted as a doctor's dissertation.[6] In February, 1925, Mr. Ettlinger visited the mine and made a detailed petrographic study of the wall rocks and vein material.

The authors desire to express their indebtedness to Mr. A. J. McNab, vice-president, to Mr. W. C. Browning, former general manager, and to

[1] F. L. Ransome: Copper Deposits near Superior, Arizona: *Bull.* 540D, U. S. Geol. Survey (1913).

[2] F. L. Ransome: Geology of the Globe Copper District, Arizona: *Prof. Paper* No. 12, U. S. Geol. Survey (1903).

[3] D. H. McLaughlin: The Origin and Occurrence of Bornite. Doctor's dissertation, Harvard University, 1917.

[4] J. C. Anderson: Application of the Zonal Theory of Primary Deposition of Ores. A. I. M. E. *Trans.* (1923) **69**, 22.

[5] W. C. Browning and F. W. Snow: Geology and Operations of the Magma Mine. *Eng. & Min. Jnl.-Pr.* (1925) **119**, 197.

[6] M. N. Short: Deep-level Chalcocite at Superior, Ariz., and at Butte, Montana. Harvard University, May, 1923.

Mr. W. Koerner, general manager, of the Magma Copper Co., for use of the mine maps, for supplying statistical data, and for permission to publish this article. They are indebted for helpful criticism to Prof. L. C. Graton, of Harvard University, under whose direction Mr. Short carried on the chalcographic work.

Summary

The rock formations at Superior are identical with those at Globe and at Ray, both of which have been well described by Ransome.[7] In the immediate vicinity of Superior, however, the Madera diorite and the Gila conglomerate are missing. The Whitetail conglomerate is present locally in small patches under the dacite.

The basal formation is the Pinal schist, of pre-Cambrian age. Overlying the schist, which was worn to a peneplain, is a great series of Paleozoic sediments comprising formations from Cambrian(?) to Carboniferous in age. The entire series presents the appearance of a conformable sequence; but at least two disconformities are known to exist.

Intrusive into the Paleozoic sediments is a great sill of diabase which in central Arizona can be traced over an area of about 1600 square miles. In the Superior district the diabase is not found in contact with formations younger than the Troy quartzite, but in the Ray district Ransome regards it as later than the Tornado limestone and assigns the intrusion to Mesozoic time. Darton, in his recently published geological map of Arizona, considers it as questionably pre-Cambrian. The illustrations of this article have followed Ransome's interpretation. If the diabase is pre-Cambrian, all the sediments in the Superior district below the Devonian limestone are pre-Cambrian, because blocks of these are found as included masses in the diabase. In the vicinity of the Magma mine the diabase invades the upper or Troy quartzite, and in the lower levels of the mine, blocks of quartzite are included in the diabase.

The authors consider that the ores of the Superior district, like those of the Miami, Ray, Globe, and Troy districts, are directly connected with the intrusion in Tertiary times of what they propose to call the Central Arizona batholith. In the Superior district dikes and sills of quartz monzonite porphyry believed to be offshoots of this batholith are intruded into both the sediments and the diabase. The ore of the Magma mine is deposited along an east-west fault, in this paper called the Magma fault. This was probably caused by crustal adjustments subsequent to the intrusion of this batholith. In part, mainly in the sediments, this fault followed the previously intruded dike of quartz monzonite porphyry but in the diabase the major influence governing the locus of the fault plane was a prior line of weakness determined by the blocks of quartzite

[7] F. L. Ransome: Copper Deposits of Ray and Miami, Ariz. *Prof. Paper* 115, U. S. Geol. Survey (1919).

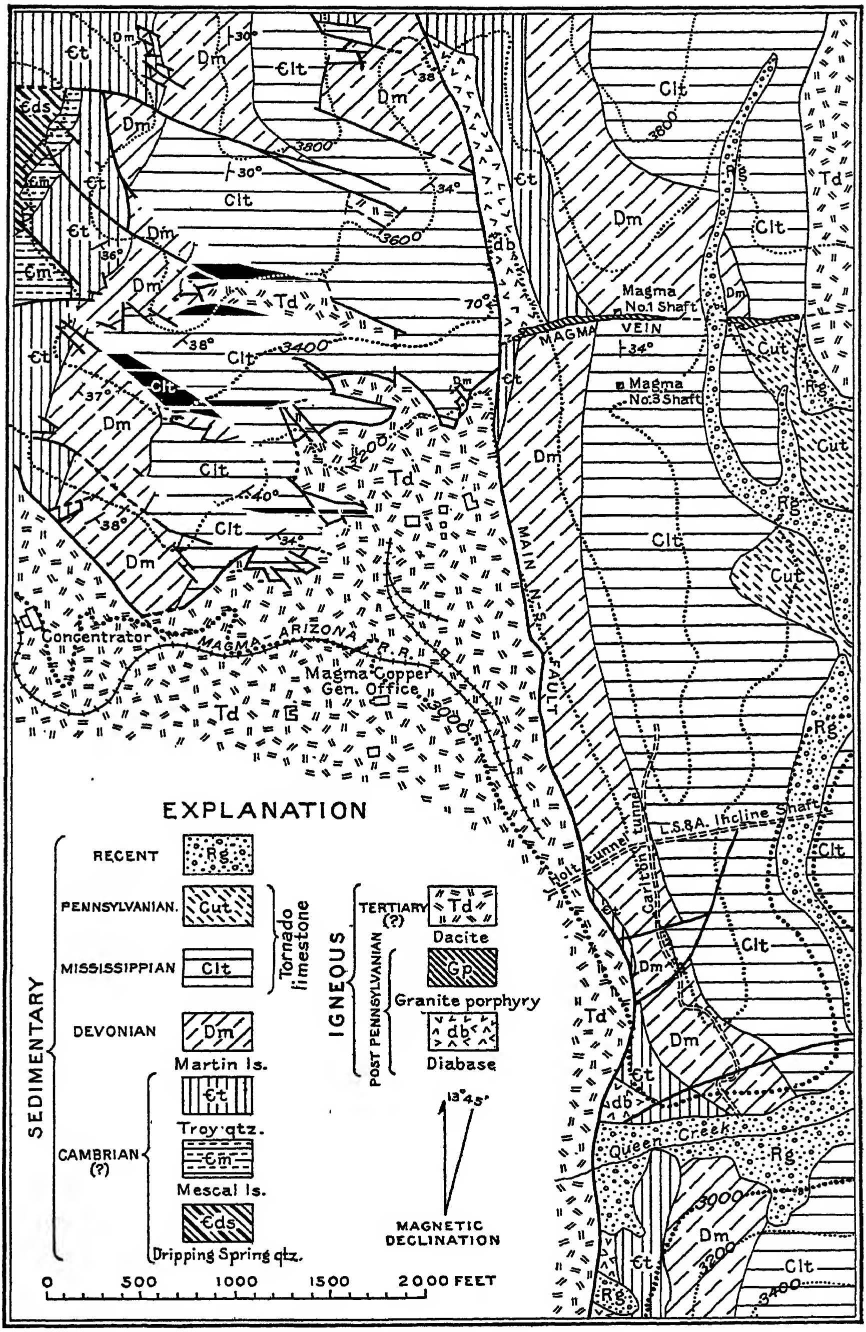


FIG. 2.—GEOLOGICAL SURFACE MAP OF VICINITY OF MAGMA MINE.

included within the diabase. Previous observers have concluded that the quartz monzonite porphyry dike was intruded into the Magma fault. More recent work in the lower levels has convinced the authors that the fault movements were subsequent to the intrusion of the porphyry.

The ore minerals occur mainly as replacements of the shattered wall rocks. In the upper levels the ores are replacements mainly of porphyry and to a minor degree of including sediments. In the lower levels most of the ores are replacements of diabase; where blocks of quartzite are found, replaced quartzite is equally important.

Following the deposition of ore there was a long continued period of erosion during which oxidation and enrichment took place. During this period the sediments and the diabase sill were approximately horizontal and the land surface was one of low relief.

Subsequent to this period of erosion and oxidation, the formations were tilted to the east approximately 35°, and the entire area was covered with dacite flows which total at least 1000 ft. in thickness. These flows covered the tilted surface of the Paleozoic formations and leveled off the entire region.

Following the extrusion of the dacite there was a period of north-south faulting which exercised the dominant control on the topographic features as they exist today. This faulting was followed by small intrusions of basalt which followed some of the fault fissures.

During the Pleistocene epoch part of the district may have been covered with Gila conglomerate, which is so well represented at Globe, Ray, and Miami. There is no record of this, however, and if the Gila conglomerate was ever present it has been removed by erosion.

Bornite is the principal hypogene ore mineral, followed in importance by chalcopyrite, deep-level chalcocite, and tennantite. Pyrite, galena, and sphalerite are abundant in places. Water was first encountered at the 400-ft. level of the Magma mine. Oxidation and enrichment are important in the upper levels of the mine. The bottom of the zone in which iron oxides occur is in general not related to the present water level but follows closely the top of the Troy quartzite.

Chalcocite and covellite are the principal supergene minerals. The 900-ft. level marks the bottom of the main chalcocite enrichment, but small seams of supergene chalcocite are found here and there as far down as the 2000-ft. level. Covellite is likewise found on the 2000-ft. level but diminishes rapidly in amount from the 900-ft. level downward. No supergene bornite was observed except as a transitional stage in the enrichment processes, and practically all is believed to be hypogene.

In places chalcocite is intergrown with bornite in graphics and similar structures indicating contemporaneity. This chalcocite is important and persists to the deepest workings attained. It is concluded that the deep-level chalcocite is hypogene. The mine is now 2250 ft. deep and

there has been no decrease of the copper content of the hypogene ores with depth.

Rocks of the District

Sedimentary Rocks

A generalized columnar section, showing the stratigraphic sequence, is shown by Fig. 3. The oldest formation of the region, the Pinal schist,

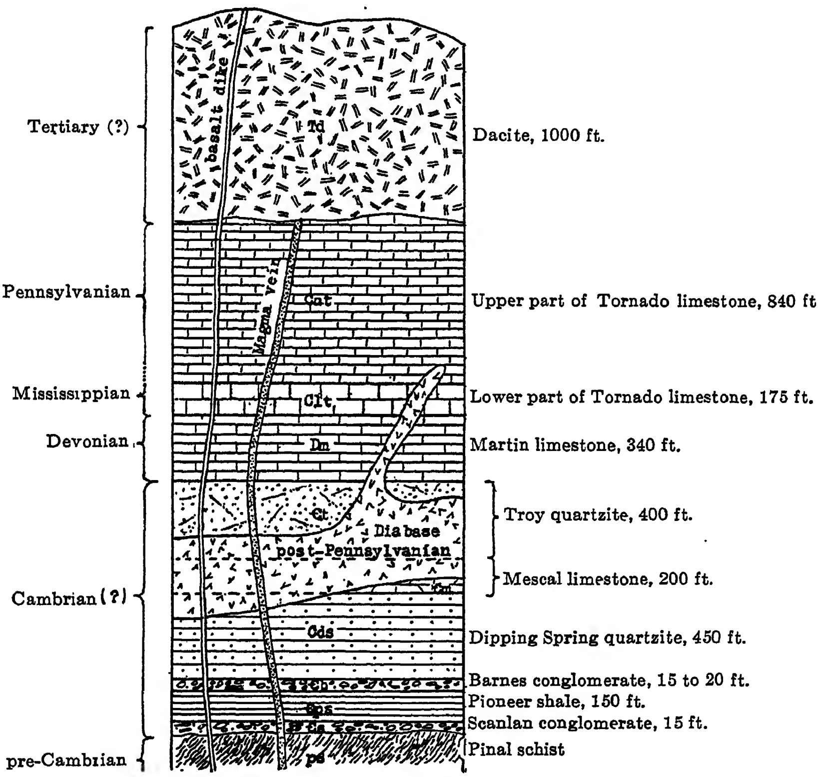


Fig. 3.—Generalized geological column of Pioneer Mining district, Arizona.

comprises a series of highly metamorphosed quartzites, muscovite and quartz-sericite schists.[8] The formation is widespread throughout central and southeastern Arizona. It is not exposed in the immediate vicinity of the Magma mine, nor encountered in any of the mine workings, but is found at Pinal 2 miles to the southwest and at Silver King 2 miles to the northwest.

[8] The descriptions of this and the Paleozoic formations have been taken in part from Ransome's report on the Ray and Miami copper districts.

Apache Group.—Proceeding upward stratigraphically from the top of the Pinal schist, the first six formations form a series known as the Apache group. An abbreviated outline is given in the geological column.

GEOLOGICAL COLUMN OF APACHE GROUP

Period	Formation	Thickness at Superior	Description
Cambrian(?)	Troy quartzite	400 ft.	Cross-bedded, pebbly beds with some thin shaly layers. Distinguished from Dripping Spring formation by lack of arkosic material.
	Mescal limestone	200 ft.	Thin cherty beds alternating with softer calcareous beds. The topmost bed is a layer of altered vesicular basalt.
	Dripping Spring quartzite	450 ft.	Upper third—thin rusty beds somewhat calcareous. Middle third—Massive even-grained buff or pinkish quartzites associated with some shaly layers. Lower third—Hard fine-grained arkosic quartzites with stripes parallel to bedding.
	Barnes conglomerate	15–20 ft.	Hard quartzite pebbles with pink arkosic matrix. Pebbles are larger than in Scanlan conglomerates.
	Pioneer shale	150 ft.	Dark reddish brown arenaceous shales with light yellow or greenish elliptical spots. These spots are due to local leaching of the pigment of the shale.
	Scanlan conglomerate	15 ft.	Pebbles of schist and white vein quartz in pink arkosic matrix. This is the basal Paleozoic formation.

None of these formations contains fossils. The Apache group has been assigned by Ransome provisionally to the Cambrian. The lower formations of the group may be Algonkian. In the working of the Magma mine the Troy quartzite is the only formation of the group that is represented, but the whole group can be seen within 3 miles of the mine.

Martin Limestone.—This formation overlies the Troy quartzite without any apparent unconformity. This is the lowest fossiliferous member of the Paleozoic formations. Fossils are abundant and are of Middle and Upper Devonian age. The top of the formation is a layer of thinly laminated shales, yellow at the surface but dark gray underground; this serves as a reliable horizon marker.

Tornado Limestone.—This formation consists of a thick series of very pure gray limestone beds. The lower 75 ft. is a cliff-maker. This is succeeded by a massive member about 100 ft. thick containing only traces of bedding planes. Above this the beds are thinner but lithologically

similar to the preceding. The lower part of the Tornado limestone contains Mississippian and the upper part Pennsylvanian fossils. The formation is 1000 ft. thick plus an unknown thickness removed by erosion.

Igneous Rocks

Diabase.—The most abundant rock encountered in the mine workings is diabase, which has been fully described by Ransome in the Globe and Miami professional papers. The diabase at Superior is undoubtedly a manifestation from the same underground source. Exposures of diabase of this same character have been observed over an area of 1600 square miles in central Arizona. In the Superior district it occurs as sills intruded into all of the pre-Devonian sediments. Its favorite locus of intrusion is the Mescal limestone, which, in many places, disappears altogether or is found as isolated patches engulfed in the diabase. Throughout the entire district the diabase has had a regional contact metamorphic effect on the Mescal limestone which is manifested by the crystallization of the limestone and the introduction of tremolite.

In the vicinity of Superior the full thickness of the diabase has not been revealed. The lower contacts where seen at the surface are fault contacts. The thickness at Superior is greater than that measured elsewhere in Gila or Pinal counties. In the Old Dominion mine at Globe the main sill is 1400 ft. thick and lies just above the Mescal limestone. Below the sill the other formations of the Apache group are found in their regular sequence.

In the Magma mine the diabase cuts the Troy quartzite and in the lower levels blocks of this formation are found in the body of the igneous rock; the upper contact between diabase and quartzite is even and shows no pronounced irregularities. (See Figs. 4 and 7.) At the contact the diabase is fine grained due to rapid chilling and here both biotite and epidote occur in the quartzite. A peculiar property of the diabase is its jointing. The joint planes are parallel to the upper surface of the sill, and since all the rocks of the mine have been tilted, the joint cracks have the same dip and strike as the overlying Paleozoic sediments. This gives the appearance of bedding. The lower surface of the diabase has not been reached by drilling or by the mine workings, but a minimum thickness of 1950 ft. is indicated by the explorations to date. Included within the diabase in the lower 600 ft. of mine workings are blocks of Troy quartzite which have the same attitude as the main body of the Troy quartzite above the sill. It is highly probable that in the Magma mine, as in the Old Dominion, the diabase is underlain by the remaining formations of the Apache group, which will be encountered if the mine is sufficiently deepened.

There is abundant evidence that the diabase has broken across the formations of the region, as north of the Magma concentrator it is included within the Dripping Spring quartzite. Part of this formation

and all of the Mescal limestone lie above the diabase, whereas in the mine the upper surface of the diabase is 400 ft. higher stratigraphically.

Freshest specimens of the diabase show tabular plagioclase laths up to 3 mm. in length surrounded by smaller augite crystals whose forms

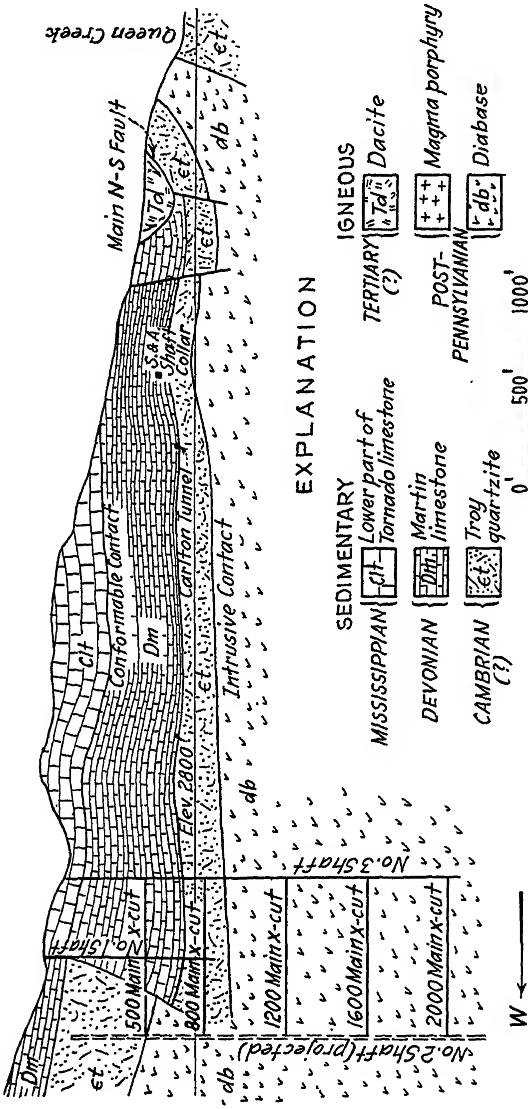


Fig. 4.—North-south geological section through No. 1 shaft.

mold themselves to the feldspar. Fresh specimens of the diabase are rare at Superior and with a very few exceptions the rock is much serpentinized and uralitized. Chlorite is abundant. In places this alteration gives the diabase an appearance of a dark organic shale. The uralitization is

probably due to dynamic causes. The extensive development of serpentine and chlorite is probably due to solutions carried by the diabase itself or proceeding from the magmatic reservoir which supplied the diabase. The widespread occurrence of this alteration precludes the possibility that it was brought by the ore-bearing solutions.

Alteration of the diabase by ore solutions can be seen in all its stages. The diabase quickly loses its dark green color. Quartz, sericite, and carbonates (calcite and rhodocrosite) are developed from the feldspars and ferromagnesian minerals. Unless the alteration is nearly complete, areas of leucoxene remain as residues from the titanium content of both the ferromagnesian minerals and ilmenite. Where both the diabase and the porphyry are highly altered, these titanium residues are diagnostic in differentiating between them.

Owing to the extensive development of serpentine and uralite, the diabase has been rendered almost impervious, and the influence of the ore solutions dies out quickly a short distance from the shattered vein zone. The diabase is known as the "tight" formation of the mine and workings in it are dry. Only locally where cut by small seams and fractures is there a small seepage of water.

Porphyry.—In the Superior district there are numerous small exposures of porphyry both as dikes and sills. Although quartz phenocrysts are comparatively rare, the rock may be classed as a quartz monzonite. The age of this porphyry is post-Pennsylvanian and is the same as that of the Schultze granite of the Miami region whose nearest exposure is 9 miles east of Superior, and that of the monzonite of the Silver King region whose nearest exposure is 3 miles north of Superior. Ransome has shown that the Schultze granite is probably of Tertiary age.[9] All of these intrusions are believed to be part of what the authors propose to call the Central Arizona batholith and with which the ores of Ray, Miami, Globe, Troy, and Silver King regions are related. It is highly probable that the quartz monzonite porphyry exposures of the Superior region are offshoots of this same batholith which underlies the whole region and were emplaced at the same time as the Schultze granite stock of the Miami region.

Dacite.—The most extensive rock of the district areally is dacite. It overlies the Paleozoic sediments and fills the inequalities of a post-Paleozoic erosion surface. Only where erosion has removed the dacite are the younger rocks revealed. The largest area of dacite in the district occupies the summit of the range and extends for many miles in its general direction. Its width is from 4 to 10 miles and its thickness is believed to be in excess of 1000 ft., although the lower contact of the formation is revealed only at its exposed edges and may extend to greater depths within the mass. Its western margin is a precipitous bluff known locally

[9] F. L. Ransome: The Copper Deposits of Ray and Miami, Arizona. U. S. Geol. Survey *Prof. Paper* 115 (1919) 59.

as Apache Leap. It is the most prominent topographic feature in the landscape. Another extensive area of dacite, which is believed to be a portion of the preceding that has been carried down by faulting, is the one which underlies the town of Superior and the flat south and west of the Magma concentrator. This dacite is lithologically similar to the dacite of Apache Leap.

The present attitude of the dacite flow of Apache Leap is apparently almost horizontal. At the summit of the range it forms the undulating plateau known as Oak Flat, which extends eastward from the Apache Leap escarpment. This plateau is cut by an exceedingly abrupt gorge known as Devil's Canyon, which, though approximately 800 ft. deep in its deepest part, apparently does not cut through the dacite flow.

FIG. 5.—HORIZONTALLY BEDDED TUFFS AT TOP OF DACITE, OAK FLAT AND NEW SUPERIOR-MIAMI ROAD.

Small patches of white rhyolite tuff may be seen along the Superior-Miami highway where it crosses Oak Flat (see Fig. 5). These tuffs are thin bedded and horizontal. It is apparent that the tuffs are simply a manifestation of the volcanic activity which gave rise to the dacite. The evidence is clear that on Oak Flat the flows are essentially horizontal.

The prevailing color of the dacite is a light pink, due to the minute flakes of iron oxide scattered through the rock. This pink color is only superficial, as fresh fractures show a much lighter creamy color. The dacite weathers into rounded boulders and blocks, giving it a rough surface difficult to traverse. The red color of the dacite contrasts markedly with the prevailing grays and whites of the underlying Paleozoic sediments.

Basalt.—Later than the post-dacite faulting is an amygdaloidal basalt which is present as a few small dikes and plug-like masses. One of these dikes fills the main north-south fault in places. Another cuts the Magma

vein in one of its richest spots but does not displace the vein. This basalt intrusion concludes the geological history of the district as now revealed.

Structural History

Crustal movements including folding, faulting, and tilting, and gneous activity have all played their part in the mountain-making processes of the region. Of these factors, faulting has been most important. Owing to their rigid character, the rocks tend to yield to deformation by breaking rather than by bending.

The mountain ranges in central and southeastern Arizona have a general northwest trend and are in the main blocked out by faults striking in the same direction. Thus the major structural features of the region are uplifted and tilted fault blocks modified by erosion. The intramontane valleys, such as that occupied by the town of Superior, represent the down-faulted portions which are filled in part by detritus removed from the neighboring mountains.

The tectonic forces causing the faults of this region are tensile stresses. The blocks have collapsed, due to the removal of support. It is logical to assume that this lack of support is due to the removal of material by the vast igneous outflows and intrusions. As a result, most of the faults are normal, that is, the hanging-wall side has dropped relative to the footwall side. Reverse faults are only of subordinate importance and, according to Ransome, are due to local compressive stresses caused by wedging between major faults.

There are two major systems of faults at Superior, an earlier east-west system and a later north-south system. The present mountain ridges have resulted from differential movements along the latter system of faults. The older east-west system is of vastly greater economic importance, as the orebodies of the region are for the most part located by these faults.

Magma Fault

The Magma fault strikes approximately east and west at right angles to the strike of the formations that it displaces. The fault belongs to the early system of faulting which was probably caused by adjustments subsequent to the emplacement of the Central Arizona batholith and of which the minor porphyry intrusions of the district are offshoots. The fault drops the sedimentary formations on its south side about 500 ft. The offsetting effect of the fault in the upper levels is described by Ransome as follows:[10] "An observer walking east on a level driven along the vein has on his right hand rocks higher in the stratigraphic column than those on his left. Moreover, in consequence of the general easterly dip of the beds, after he first sees limestone on the right hand,

[10] F. L. Ransome: U. S. Geol. Survey, *Bull.* 540 (1912) 145.

he must continue for 400 to 450 ft. before he reaches the same stratigraphic horizon on the left."

The movement on the fault plane probably did not take place as a quick vertical break but was a slow differential movement in which each wall took part at different intervals. The movement was not a vertical drop but one oblique to the fault plane. Fig. 6 is offered as a suggestion to indicate the direction of movement. East of the shafts the bend in the Magma vein occurs at a higher elevation than at the shafts, as indicated in the stereogram. The line joining the bend in the vein at

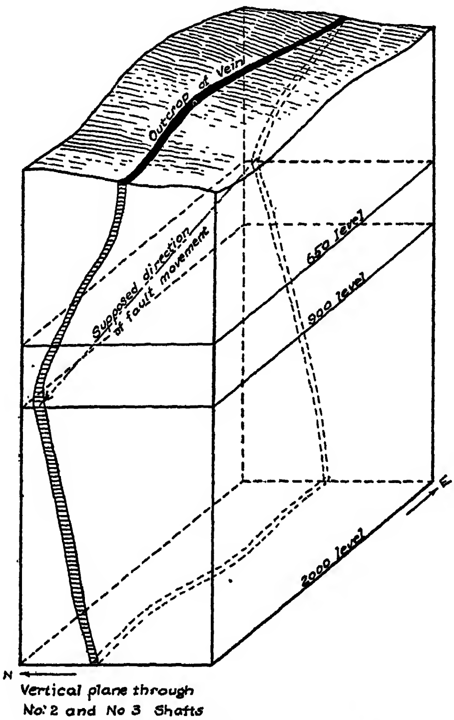


FIG. 6.—STEREOGRAM OF MAGMA VEIN.

the 650 and 900-ft. levels is the direction of the fault movement. The formations on the south side of the fault have been moved relatively downward and to the west of those on the north side. The movement is more nearly horizontal than vertical.

The reasons for this assumption are apparent by an inspection of Fig. 6. Assuming the movement vertically downward, the formations above the 900-ft. level to the north of the fault could not fit snugly to those south of the fault without great shattering and the formation of subordinate fractures. In general little or no timber is required in crosscuts away from the fault itself. In places the fault is relatively clean

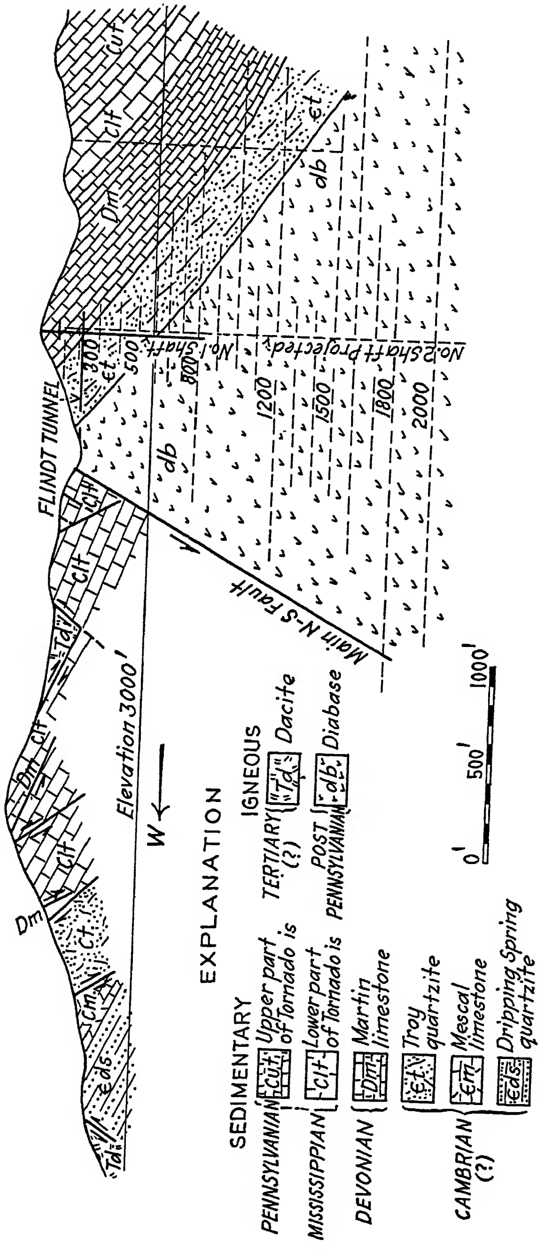


FIG. 7.—EAST-WEST GEOLOGICAL SECTION THROUGH SHAFTS.

and sharp and in the absence of shattering difficult to follow. The indicated direction of movement is only approximate as the Magma fault is a warped surface and it is not likely that the axis of the bend is a straight line.

In addition to the Magma fault there are in the district several other minor fractures or faults of east-west trend which show mineralization but to date their development has not proved satisfactory.

Post-dacite Faults

The major faulting of the district is later than the dacite flows and consequently is of Tertiary or younger age. The main north-south fault is the most important structural feature of the district. It strikes

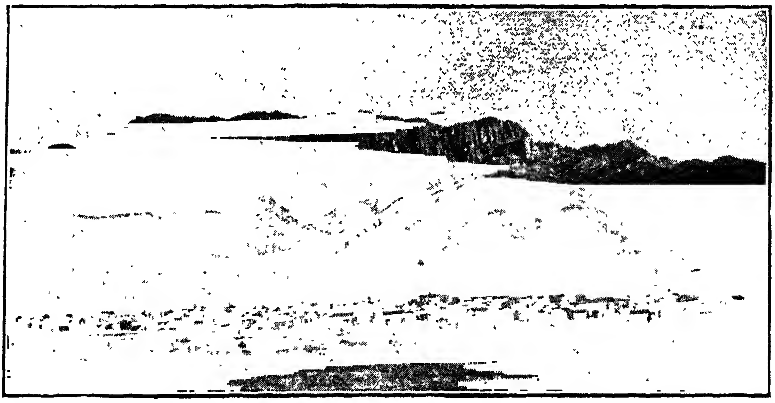


FIG. 8.—APACHE LEAP AND SUPERIOR, ARIZ., LOOKING EAST FROM RAILROAD STATION.

approximately parallel to the general trend of the range and dips westerly 65° to 70°. Underground it is cut by the main adit (500-ft.) level, 1200 and 1800-ft. levels of the Magma mine and by the Holt tunnel of the old Lake Superior and Arizona mine. In the main adit level it is represented by a crushed zone 10 to 20 ft. wide. Dacite occupies the hanging wall and Troy quartzite the foot wall. As the surface of the fault is inconspicuous and as it lies at the foot of the somewhat precipitous ridge (Fig. 8), it is for the most part covered with desert wash, and good exposures are infrequent. Although the fault breccia is indurated, the fault does not outcrop above the surrounding surface as do many others of far less importance in the region.

The length of the main north-south fault is considerable. It has been followed 2 miles north and 4 miles south of the Magma mine. The total displacement of the fault is not definitely known. To the north it seems to split up into two branches. The vertical element of displacement increases to the south; thus the fault is rotatory. Near the town of Superior the difference in elevation measured by the top of the dacite on

Apache Leap and that in the flat is over 1000 ft., and the displacement is probably at least as great. In the vicinity of the mine, one mile to the north, the vertical displacement is approximately 600 ft. in the sediments north of the Magma fault. There has been a large horizontal component of displacement as well as rotation. The formations west of the main north-south fault have been moved relatively to the south of those east of the fault.

East of the main north-south fault the geology is comparatively very simple. The sediments dip to the east about 30°, but with the exception of the Magma fault and some less important east-west faults the beds are practically unbroken. West of the main north-south fault a far different state of affairs exists. Most of the sedimentary formations have been covered by dacite and the structural relations concealed, but here and there the dacite has been stripped off by erosion. The limestone hill between the mine and the concentrator is a veritable mosaic of fault blocks which trend in every conceivable direction. Here and there small patches of dacite in the down-faulted blocks remain, indicating that the faults are post-dacite and presumably of the same age as the main north-south fault.

The attitude of the beds in these blocks varies from place to place, but the strike averages from N 60° E and the average dip is south 40°. Since the beds east of the main north-south fault strike north and south and dip 30° to the east, it is seen that the attitude of the beds west of the main north-south fault is at a wide variance with that of the formations to the east of the fault. Undoubtedly there has been great shattering on the hanging wall of the main north-south fault in consequence of a movement of such a magnitude, but the causes of such a wide variance in the attitude are entirely unknown. Fig. 7 is misleading in that it shows only apparent dips and does not record this wide variation in the strike of the fault blocks.

Mineralization

The principal copper-bearing hypogene minerals are bornite and chalcopyrite, the former predominating. Chalcocite, covellite, and the oxidized copper compounds are the dominant ore minerals above the 800-ft. level where enrichment has been most effective. It is held that both the ore solutions and the porphyry came from the same igneous source at somewhat different time intervals but in the same geologic cycle. The ore solutions followed the east-west faulting which was subsequent to the intrusion of the porphyry. This fault zone was the line of least resistance and was much more permeable than the porphyry dike. This is evident because in the lower levels the porphyry is absolutely barren and unshattered and is entirely outside the fault zone. The mineralized fault zone is here more or less parallel to the porphyry and a few hundred

feet from it. No porphyry occurs in this zone and the ore is replaced diabase and quartzite. However, in the upper levels, in the sediments

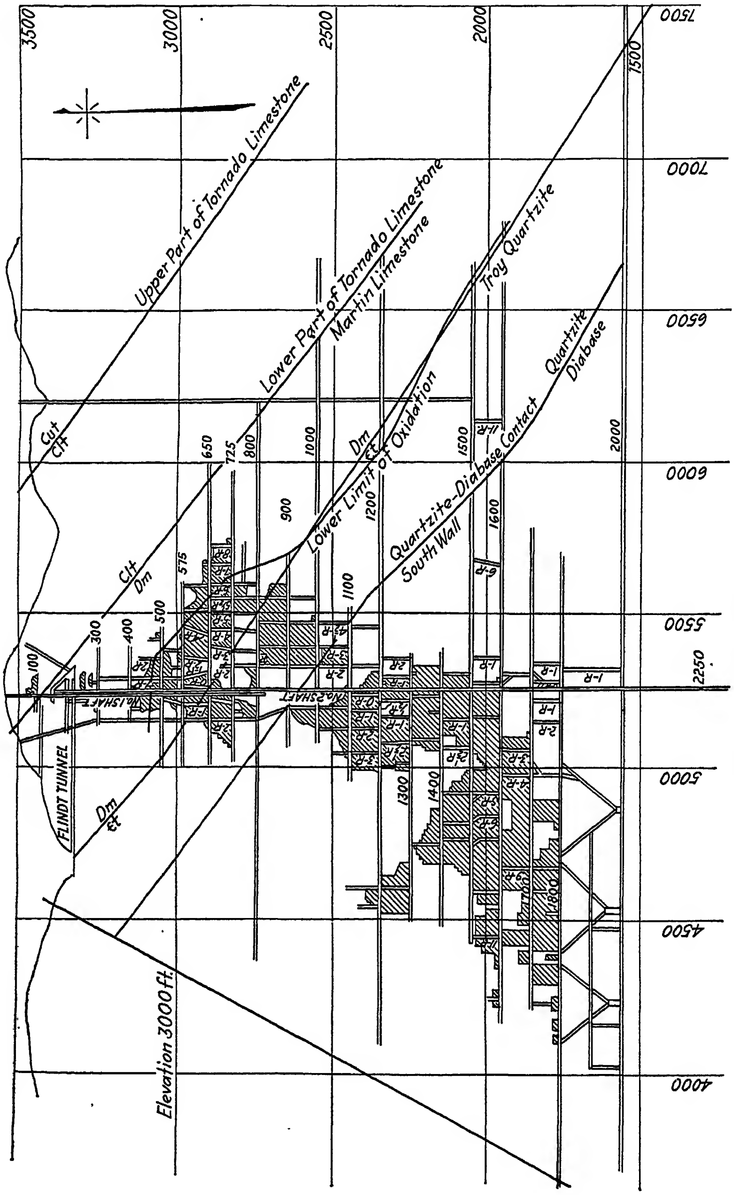


FIG. 9.—EAST-WEST LONGITUDINAL PROJECTION OF MINE WORKINGS.

where the fault zone followed the porphyry dike, the porphyry was replaced along with the other wall rocks.

The mineral association of the Magma ores is very similar to that of mines well out in the intermediate vein zone in Butte. Polished sections of ores from the Snowball vein of the North Butte mine containing bornite, chalcopyrite, chalcocite, sphalerite, galena and tennantite are indistinguishable under the microscope from sections of Magma ores. The Magma vein resembles the Snowball vein in that it has only a small contact effect on the surrounding rocks. The Magma and Snowball ores were evidently deposited under similar conditions of temperature and pressure.

There is no evidence that the wall rocks of the Magma vein had any influence on the initial mineralization. In other words, the ores are the replacement of whatever wall rocks of the fault zone came in contact with the ore solutions. The character of the ores that replaced quartzite is identical with that of the ores that replaced diabase. Most of the ore mined to date has been replaced diabase because diabase happened to be the most abundant rock in the fault zone traversed by the ore solutions. In the levels below the 1600-ft. level, where the fault zone passed through blocks of quartzite included in the diabase, replaced quartzite is equally important with replaced diabase as ore.

Fig. 9 shows the vertical longitudinal projection of the ores already mined. The hatched areas indicate the stopes as of Oct. 1, 1924. Below the 1600-ft. level the orebody rapidly increases in size and the lowest level, 2250 ft., shows as much ore as any level above and with the grade equally good if not better. An inspection of Fig. 9 shows that there is a pronounced pitch of the orebody to the west. The axis is nearly at right angles to the dip of sediments. It will be shown later that the tilting has taken place since oxidation of the orebodies. Hence there is good reason for believing that the axis of the orebody stood vertical at the time of its formation.

Zonal Distribution of Ores

An elaborate theory of zonal hypogene ore deposition in the Magma mine has been proposed recently by John C. Anderson.[11] According to Anderson, the zones of deposition are parallel to sedimentary formations and more or less coincide with them. The uppermost zone is a zone of manganese and silver ores with a little overlapping copper. This coincides with the limestones. Coinciding with the quartzite and extending below it is a zone of supergene enrichment and zinc ores carrying silver. At 500 ft. stratigraphically below the quartzite all the zinc minerals are bottomed. Here the vein averages 8 ft. wide and carries about 5 per cent. copper with a little silver. Below that point there is a steady increase in the size and value of the vein until at the 2000-ft. level

[11] John C. Anderson: Application of Zonal Theory of Primary Deposition of Ores: A. I. M. E. *Trans.* (1923) **69**, 22.

it averages 20 ft. in width and carries more than 8 per cent. copper. Since the formations are tilted, the zones are also tilted. Hence one starting at the main crosscut (the crosscut joining No. 2 and No. 3 shafts) on the 1200-ft. level would be in a zone of intense copper deposition. As he proceeds eastward he passes successively through a zone of supergene copper enrichment with some hypogene lead and zinc ores, then the zone of oxidized ores, and finally into a zone of manganese and silver ores. According to the scheme proposed by Anderson where manganese is encountered, copper minerals will be found at depth. Since all the formations of the district are tilted to the east, the effect of depth will be attained by proceeding westward on the same level.

We are in agreement with Anderson regarding the principle of zoning, but are not in accord with his ideas of its widespread application. Moreover, his description and sketches are open to considerable modification. The Magma mine was worked for silver in the early days; this silver, however, was not associated with manganese but with chalcocite. The silver taken from these operations was not important. It was surely the result of supergene enrichment and not of hypogene deposition. The zone of manganese and silver ores as depicted by Anderson has no existence.

According to Anderson, the zone of supergene enrichment is just above the zone of hypogene lead and zinc deposition. The inference is that there was no hypogene copper sulfide deposition in this zone and above it. Our work with the microscope convinces us that the supergene copper ores are mainly the result of the enrichment of almost equally rich bornite ores which extended practically to the surfaces. The oreshoot was smaller than on the lower levels but equally rich, if not richer. If lead and zinc existed on the upper levels they have been completely replaced by supergene copper sulfides.

We believe that zoning is attested by the presence of large quantities of lead and zinc on the eastern margin of the main oreshoot from the 1000-ft. level to the lowest levels reached. This does not signify that lead and zinc are not present in the central portions of the orebody, as here and there pockets of considerable size are occasionally found. Both sphalerite and galena are found in microscopic amounts even in the apparently purest bornite ores.

Work with the microscope establishes the fact that below the zone of enrichment there is no change in the hypogene minerals themselves or in their relations to each other. That is, intergrowths such as those between bornite and tennantite in ores from the 2000-ft. level have exactly the same appearance as corresponding intergrowths in ores from the 1000-ft. level. Such changes as are noted are changes in amounts of these minerals. It is doubtful within these limits whether there is much difference in the proportions of the copper minerals to each other.

Oxidation

Considering the large proportion of iron in the Magma ores and their concentrated rather than disseminated nature, it is surprising that the outcrop does not show a strong gossan. The oxidizing process has resulted in a leaching of iron at the surface (Fig. 10).

Fig. 11 shows an outcrop of the Magma vein which is indicated by the line of prospect pits.

At a moderate depth, about 100 ft. below the surface, carbonates of copper with only a relatively small proportion of iron oxides appear.

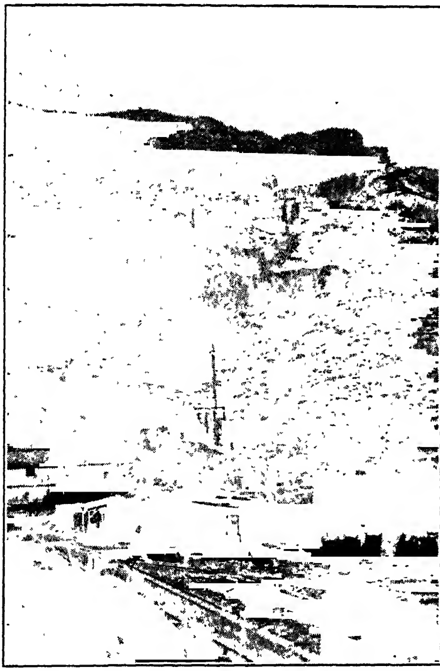


Fig. 11.—Outcrop of Magma vein shown by line of prospect pits, from portal of Flindt tunnel.

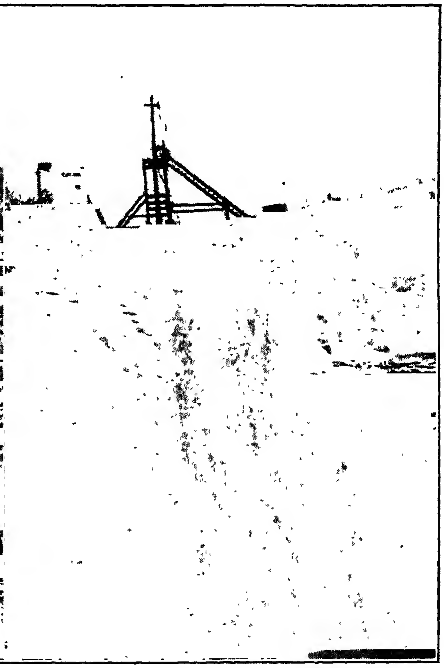


Fig. 10.—Outcrop of Magma vein, glory hole and No. 1 shaft. Observe porphyry dike (white) contrasting with limestone (dark).

Even at the top of this zone the process was incomplete, and pockets of chalcocite are found here and there marooned high above the general zone of supergene enrichment. The relatively impermeable character of the ores themselves has without doubt been a factor in protecting them from complete oxidation, but the chief influence has been the extremely arid climate of the region.

The absence of a gossan is attributable to the extremely low proportion of pyrite in the hypogene ore in the present outcrop. Blanchard and

Boswell state that hydrolysis of ferric iron to limonite is prevented by free sulfuric acid and promoted by copper.[12] In other words, a high concentration of copper in a ferric sulfate solution is held to counteract the effect of sulfuric acid in keeping the iron in solution and to precipitate the iron as limonite. Pseudomorphs of limonite after pyrite are most commonly formed by the neutralization of H_2SO_4 by carbonate solutions. It has already been stated that the proportion of copper to iron in the Magma ores is unusually high. It follows that if pyrite had been present in a small or moderate amount in the present outcrop before oxidation took place, a considerable part of the iron would have remained in the vein as limonite.

Furthermore, Blanchard and Boswell show that sericitized porphyry acts similarly to carbonates in that it tends to neutralize sulfuric acid and thus permit ferric iron solutions to hydrolyze.[13] Oxidation products from pyrite in general travel only a limited distance in sericitized porphyry before being precipitated. Therefore had there been a small or moderate amount of pyrite in the present outcrop some of this iron would have precipitated in the porphyry as limonite.

Following Blanchard and Boswell, the term "limonite" is used as a field name for fine-grained yellowish or brownish deposits from decomposing iron-bearing minerals. Presumably the limonite is usually ferric oxide monohydrate (goethite), but it may be ferric oxide (hematite), lepidocrocite, jarosite or certain basic ferric sulfates.[14]

When extensive development was undertaken about 1910, the water level stood at about the 400-ft. level. It was recognized when exploration had been carried somewhat deeper, that the zones of oxidation and enrichment had no apparent relation to the present water level. Sulfides prevail in the western portions of all the levels of the mine from the 500-ft. to the 800-ft. In the extreme eastern parts of these levels carbonates and oxides of copper appear with the sulfides, and the porphyry is iron-stained where not mineralized. Above the 500-ft. level the bottom of the oxidation zone is practically horizontal.

Below the 800-ft. level the zone of oxidation does not reach the productive ores of the mine, but at a certain distance successively farther to the east in following the vein downward, the porphyry changes in character from a grayish white color with little or no iron oxide to a rusty brown or gray color with reddish brown streaks here and there along fracture planes. Similar changes are noted where the vein lies entirely within the diabase. The line where this change occurs is the

[12] R. Blanchard and P. F. Boswell: Notes on the Oxidation Products Derived from Chalcopyrite: *Econ. Geol.* (1925) **20**, 616, 1925.

[13] *Op. cit.*, 631.

[14] *Op. cit.*, 615.

lower limit of intense oxidation, indicated in Fig. 9. Although the elevation of the surface increases toward the east, the bottom of the oxidation zone shows no parallelism with the present surface but dips in the opposite direction into the hill.

The local base level of the region is the bed of Queen Creek and the drainage is to the west. Moreover, the water level, before it was lowered by pumping, stood 150 ft. above the local base level and instead of being lower to the east was actually higher. Experience has shown that important oxidation below the water level is exceptional without long periods of geological time or unusual conditions of water circulation. And even with these conditions it is apt to be very local and to be manifested as cusps extending down from a more or less regular surface.

Could the oxidation have taken place after tilting? If so, the water level must have been at least 1200 ft. deeper than at present, as oxidized ores are present in the 1600-ft. east drift and the present water level as already mentioned is the 400-ft. level. It is more probable that in the period which has elapsed since the beds were tilted, it stood much higher than at present. The thick flows of dacite would tend to elevate the water level. Since the formation of the main north-south fault erosion has been at work and the water level is gradually being lowered.

The striking parallelism shown by the bottom of the zone of oxidation and the dip of the sediments strongly suggests that the oxidation took place previous to tilting; in other words, when the attitude of the beds was essentially horizontal.

Another feature of interest is the confinement of intense oxidation to places in which the vein is in Paleozoic sediments. Little limonite or other oxidation products have been found where the vein is in diabase. This is believed to be due to the differences in permeability. The diabase away from the fault zone is very impermeable in comparison with the sediments. The permeability of the sediments permits an unimpeded general circulation, but below the diabase contact, water circulation is limited to the fault zone itself. It is reasonable to suppose that within the vein circulation was very limited; quoting Lindgren's description of underground waters at Cripple Creek, the shattered wall rocks of the fault zone acted as "a sponge in a cup." The circulation was thus too limited to permit oxidation but was sufficient to produce small quantities of supergene copper sulfides.

Summarizing the evidence of oxidation, it occurred in the period following ore formation and before the sediments were tilted. The sediments were horizontal and the bottom of the oxidation zone was undulating and more or less parallel to the surface. The depth of oxidation was at least 1500 ft. and probably much greater, as the top of the Pennsylvanian limestone has been eroded off. Such a depth of oxidation is strongly indicative of an arid climate at that time.

Referring to Fig. 9 again, it will be noted that above the 500-ft. level the lower limit of the zone of oxidation is essentially horizontal and is only 50 ft. below the pre-mine water level. This would indicate that it was determined by recent oxidation. In relatively recent times the water level could have been 50 ft. lower than at present, and the zone of oxidation could have followed it to this depth. The dacite has been removed from this portion of the deposit in relatively recent times, and it is logical to assume that recent oxidation would be effective.

Thus, for this portion of the mine there have been two periods of oxidation; recent oxidation has been superimposed on an earlier oxidation.

Paragenesis of Ore Minerals

Criteria of Replacement

The study of the paragenesis of the sulfides was made on a suite of 230 polished sections. This study was made for the purpose of establishing the hypogene or supergene origin of the deep-level chalcocite; incidentally, it confirmed the hypogene nature of the bornite which was earlier established by D. H. McLaughlin.[15] The hand specimens from which the sections were cut were collected by members of the Secondary Enrichment Investigation in 1916 and 1917, and by the authors in 1921.

Although the study of mineral paragenesis does not result in the direct discovery of new orebodies, it is of value in a general study of an ore desposit in that it gives a clue to the complex processes of ore deposition and to the character of the ore solutions. A study of paragenesis is indispensable in determining the hypogene or supergene nature of sulfide ores.

The best evidence of the replacement of one mineral by another is afforded by veinlets or indentations of one mineral in the other. The evidence is still stronger if the later deposited mineral follows a crack or cleavage direction in the earlier. If an apparently unbroken crystal of one mineral is surrounded by another the evidence is indecisive. Garnet and pyrite in schist are of later origin. On the other hand, pyrite crystals with apparently uncorroded faces have been observed in the midst of other sulfides, although the weight of evidence which has accumulated makes it certain that pyrite is one of the earliest if not the earliest sulfide to deposit in any sulfide orebody. The most probable explanation of pyrite cubes surrounded by other sulfides is not that the cubes are later than the surrounding minerals, but that they formed in some earlier gangue mineral which was replaced by later sulfides in preference to pyrite. The same reasoning can in some cases be applied to veinlets of one mineral cutting another. In some of the Magma polished

[15] D. H. McLaughlin: The Origin and Occurrence of Bornite. Doctor's dissertation, Harvard University, 1917.

sections veinlets of quartz cut bornite. At first sight one would conclude that quartz is later than bornite. By careful search with the higher powers of the microscope these quartz veinlets show embayments and indentations pointing inward. There has been some corrosion of quartz by the solutions which deposited bornite. The quartz veinlet cut across altered diabase and was resistant to replacement by ore solutions, whereas the diabase was completely replaced.

It is not inconceivable that when a well formed crystal of one mineral is surrounded by a second mineral, both are of the same age. The mineral having the weaker tendency to form crystal faces is forced to accommodate itself to the crystal planes of the mineral having the stronger tendency.

If the pattern made by two adjacent minerals is smooth and regular or forms regular curves with no decided projections of one mineral into another it is termed the "mutual boundary pattern" by Graton and Murdoch, who first described it.[16] This pattern has been almost universally held to be produced by the contemporaneous deposition of the two minerals; neither mineral is able to assume its characteristic crystal form, but growth proceeds outward from nuclei of the two minerals respectively, and each mineral grows by accretions at its exterior surfaces until the minerals join. Wandke states: "The chalcocite-bornite mutual boundary pattern is shown to have been produced in one case by mutually interfering growth. A large trapezohedron of bornite from the Alps was contributed for study by Professor Palache. The bornite crystal had evidently grown in a vug attached to calcite. On several faces there were small orthorhombic crystals of chalcocite variously oriented. When ground down and polished the bornite crystal showed the boundary between the bornite and chalcocite to be of the mutual character.[17] Another pattern made by two minerals shows indentations or cusps of one mineral projecting into the other (Fig. 29).[17a] In such patterns one is unable to decide which is the earlier mineral, and the impression of essential contemporaneity is strong. The last mentioned structure merges by a decrease in angularity into structures approaching the graphic. The discussion of the graphic structure is given later in this paper, as the conclusion regarding the origin of deep-level chalcocite hinges largely on its interpretation.

Criteria indicative of supergene replacement are:

1. Veinlets following cracks or other open spaces.

2. Replacements following boundaries between sulfides and quartz or other gangue minerals.

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[16] L. C. Graton and J. Murdoch: The Sulphide Ores of Copper. A. I. M. E. *Trans.* (1913) **45.**

[17] Alfred Wandke: The Habit, Etch-structure and Geologic Significance of Chalcocite. Unpublished abstract read before the Amer. Inst. of Mining & Met. Engrs., Feb., 1924.

[17a] All microphotography, Figs. 12 to 29, are at the end of the text.

These criteria were first proposed by Laney[18] and have stood the test of time as absolutely indicative of supergene origin of the replacing mineral. Minerals showing these structures have invariably been found to give out in depth. These two ways in which enrichment occurs do not exhaust the patterns formed by enrichment but are by far the most important. By no means all of the supergene veinlets show visible cracks under the microscope; by a careful search, however, cracks can often be observed following the veinlets for short distances.

It is not always possible to distinguish veinlets of ore minerals which are of hypogene replacement of sulfides from similar veinlets of supergene replacement. The supergene veinlets tend to be more continuous and more ramifying. Under low powers the contact between the supergene mineral in the veinlet and the hypogene host mineral is usually smooth. With higher magnifications, however, the supergene veinlet shows a distinctly serrate boundary against the hypogene host with thornlike and spinelike projections into the latter. Hypogene veinlets lack these irregularities. The hypogene veinlet also usually projects into the replaced sulfide without entirely cutting across it. Its sides are usually not parallel.

Hypogene Sulfides

Pyrite

The most abundant sulfide in the Magma mine is pyrite. This is not apparent at first sight since the ores are usually rich in copper. However, there are parts of the Magma vein where pyrite is 10 to 20 times as abundant as the copper sulfides and where the vein material is not commercial. Pyrite accompanies all the other sulfides, but where the pyrite content is high, the copper content is nearly always represented by chalcopyrite. On the other hand, where bornite constitutes a large proportion of the copper minerals, the pyrite content is usually low. This fact is explained by the relative concentration of iron and sulfur with respect to copper in the ore solutions.

Pyrite crystals usually show under higher magnifications, embayments and indentations. These indicate corrosion by solutions depositing the other sulfides. Pyrite in the "exploded bomb structure" similar to that depicted by Graton and Murdoch[19] is rather common in Magma ores. This indicates that pyrite is earlier than the minerals in the cracks of the bomb. In no case has pyrite been observed in veinlets or other structures indicating replacement of the other sulfides. The conclusion is

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[18] F. B. Laney: The Relation of Bornite and Chalcocite in the Copper Ores of the Virgilina District of Virginia and North Carolina. *Econ. Geol.* (1911) **6**, 405.

[19] L. C. Graton and J. Murdoch: The Sulphide Ores of Copper. A. I. M. E. *Trans.* (1913) **45**, 37.

that pyrite is the earliest sulfide deposited. In no instance does it appear to be later than any of the other sulfides. The impression is gained that all the pyrite was deposited before any of the remaining sulfides began to form, and there is no suggestion of an overlapping of the period of deposition of pyrite on that of any of the remaining sulfides. This does not imply that the other metals were not present in solutions depositing pyrite. No doubt they were present, but the excess of iron and sulfur was precipitated out of solution in advance of copper, lead and zinc compounds.

Pyrite is generally accompanied by some quartz, which in places forms acicular crystals in the midst of sulfides. Such crystals usually indicate vuggy mineralization. However, acicular crystals are not in themselves proof of open-space deposition as the strong tendency of quartz to form crystals may make possible their formation by replacement. Another possible explanation is that solution has gone on a little faster than deposition. There are no other evidences of open-space deposition in Magma ores. The age relationship of quartz is well established in these ores. In places quartz and pyrite show mutual boundary structures; more rarely pyrite is cut by quartz veinlets. These facts indicate that the deposition of quartz began with that of pyrite but continued after all the pyrite had been deposited. The time of maximum deposition of quartz was the period between the deposition of pyrite and of the later sulfides.

Hypogene Chalcopyrite

The most widespread copper sulfide is chalcopyrite. It accounts for almost if not quite as much copper as bornite. In the richer orebodies bornite predominates over chalcopyrite, but, on the other hand, in the low-grade pyritic ores and in zinc-lead orebodies chalcopyrite is always found in greater or less amount, while bornite is wanting or present in very small amounts. Wherever bornite ores occur, more or less chalcopyrite is associated with bornite. Mutual boundary structures and subgraphics are common. These patterns indicate that there is no essential age difference between chalcopyrite and bornite.

Bornite

In the early days of the mine the opinion was general that much bornite was formed as a supergene enrichment of chalcopyrite and consequently that the rich bornite would give way in depth to leaner pyrite-chalcopyrite ores. The presence of bornite in undiminished quantities to the bottom level of the mine demonstrates conclusively its hypogene origin. Bornite is also found in deep levels in other districts. Microscopic evidence confirms this conclusion. The intergrowths between bornite and chalcopyrite, as mentioned in the preceding para-

graph, indicate essential contemporaneity. There is some overlapping of bornite deposition on that of chalcopyrite, as attested by veinlets, cusps and indentations of bornite projecting into chalcopyrite.

Thin sections of rich bornite ores show bornite containing remnants of unreplaced rock minerals, bornite embayments in gangue, and bornite veinlets cutting across sericite and penetrating between sericite plates. Either bornite has extensively replaced rock minerals, or it has replaced pre-existing sulfides which have replaced rock minerals. In rich bornite ores the proportion of pyrite and sphalerite is very small. Although pyrite and sphalerite appear invariably to be earlier than bornite as shown by embayments in them filled with bornite, the embayments are not deep, and the original crystal outlines are often not completely obliterated by the corrosion due to solutions depositing bornite. There are no patterns such as those produced by an extensive replacement of pyrite by chalcocite (Fig. 15), in which only isolated remnants of former pyrite grains remain. In the absence of such patterns it is safe to conclude that there has been no extensive replacement of pyrite and sphalerite by bornite. Chalcopyrite is abundant and is of the same age; galena is also contemporaneous with bornite. The conclusion is justified that the amount of bornite deposited as a replacement of earlier hypogene sulfides was small in comparison with that deposited as a replacement of rock minerals.

Bornite crystals have not been observed in the Magma mine. In general bornite has a relatively weak tendency to form crystal outlines even in open spaces. It is probable that some bornite was deposited as a filling of joints and seams in the porphyry, but the formation of crystal faces would probably be prevented by the mutual interference of growing crystals.

Tennantite

The presence of tennantite in the Magma mine has not been previously described. Ransome does not mention it, which is not surprising in view of the fact that it does not occur in the upper levels. It may have been present before enrichment took place and later have been obliterated by processes of oxidation and enrichment; or its absence may be indicative of hypogene zoning. The absence of lead and zinc minerals in the upper levels is ascribed to the same cause or causes. It occurs in abundance on the 900-ft. level and below that level it continues to the 2250-ft. level in about the same proportion to the other sulfides. Tennantite is very closely associated with bornite, and no specimens which were apparently pure tennantite failed to show mutual boundary tennantite-bornite intergrowths under the microscope (see Fig. 29). The impression left by these microscopic structures is that tennantite and bornite are of the same age. Tennantite usually predominates in these intergrowths and gives

the hand specimen its characteristic gray color. The miners readily recognize it and call it "gray copper."

Sphalerite

The occurrence of sphalerite and galena has been described under the heading "zonal distribution of ores." Sphalerite is always associated with galena, and some chalcopyrite is practically always present in greater or less amount. The universal association of sphalerite and galena would immediately suggest that the two minerals are of contemporaneous deposition, but detailed investigation reveals the fact that sphalerite is usually earlier than galena as evidenced by embayments in the sphalerite. The universal association of sphalerite and galena indicates similar conditions of solution and deposition. It appears probable, however, that the sphalerite deposited first, and that it is more or less unstable in the presence of the remaining solutions that deposit galena. Sphalerite has a tendency to form crystal faces, and euhedral forms surrounded by other sulfides are not uncommon. These crystals usually show embayments due to corrosion by solutions depositing the remaining sulfides. This evidence makes it certain that the age of sphalerite is between that of pyrite and the remaining sulfides.

Galena

Some of the galena seems to be a trifle later than the copper sulfides but the mutual boundary structures indicate that most of it was deposited at the same time. Graphic intergrowths between bornite and galena are not uncommon. Such intergrowths surely indicate a hypogene origin for both minerals.

Enargite

Only one of the specimens collected by the authors contains enargite. It is evidently rare in the Magma mine. The conditions of deposition were in general such as to favor the precipitation of the arsenic as tennantite rather than as enargite.

Supergene Minerals

Oxidized Minerals

Oxidized copper ores are not abundant in the Magma mine and always contain some residual supergene copper sulfides. Supergene sulfides occur within 50 ft. of the outcrop but persist below the lowest levels in which oxidized minerals occur. The oxidized copper minerals are malachite, cuprite and chrysocolla. Small bunches or seams of chrysocolla are found here and there in the vein from the outcrop to the 500-ft. level but nowhere constitute commercial ore. A little cuprite is associated

with malachite and chalcocite in the oxidized zone. It owes its formation to the direct oxidation of chalcocite. A little native copper is observed but its occurrence is rare. Malachite is the principal oxidized copper mineral. Most of it replaced chalcocite and covellite.

Limonite occurs in distinct veinlets cutting the sulfides and gangue minerals, as brown stains in the gangue minerals, and as pseudomorphs after pyrite. The first two modes of occurrence are due to the transportation and subsequent hydrolysis of iron sulfate solutions. In the case of the pseudomorphs after pyrite, the oxidized iron remained within the boundaries of the sulfide parent. Limonite veinlets of microscopic size are found down to the 1000-ft. level but the total amount of limonite at this depth is insignificant.

Supergene Chalcocite

Massive chalcocite orebodies were mined from the 500-ft. to the 600-ft. levels. The derivation of part of this chalcocite from bornite is proved microscopically by occasional remnants of bornite in areas of chalcocite and by the fact that these rich chalcocite ores change in depth to ores in which bornite predominates. The chalcocite of the upper levels is steely-white in color in the hand specimen. Under the microscope it sometimes exhibits a peculiar mottled appearance. It is usually of fine granular texture, the different grains having different colors varying from white, through grays to very light blue. Rounded hazy patches of slightly grayer color are observed in the chalcocite and are believed to represent the "ghosts" of the replaced bornite grains, the difference in color being assumed to be due to incomplete removal of iron in the replacement process. These differences in color instantly disappear when the chalcocite is etched with an oxidizing reagent such as nitric acid or ferric chlorite; the whole area turns a dark blue.

The completeness with which the hypogene sulfides of the upper levels have been replaced by chalcocite has obliterated all but insignificant traces of these minerals. Only pyrite appears to have survived the process and even it has been strongly attacked. Areas of unreplaced bornite of microscopic size surrounded by gangue can be observed. They owe their preservation to the protective action of the gangue. A small proportion of covellite is found in steely chalcocite. The covellite has originated as a direct oxidation product of the chalcocite. The steely chalcocite zone is irregular in shape and variable in thickness. It grades gradually upwards into a zone of oxidized ores containing pockets of residual sulfides. The lower limit of the steely chalcocite zone is more regular and parallels the dip of the sedimentary beds.

Native silver is found in supergene chalcocite. It seems to be later than the chalcocite. It has not been observed in polished sections of the hypogene ores, although analyses show the presence of silver. The

hypogene silver may be present as stromeyerite or it may be contained in tennantite.

The zone of massive chalcocite changes rather abruptly in depth to a zone of rich bornite ores containing more or less supergene chalcocite. This zone is very limited and does not extend more than 150 ft. below the bottom of the massive chalcocite zone. The 900-ft. level marks its bottom at its eastern end and it extends diagonally upward toward the west, paralleling in a general way the dip of the formations. The bottom of this zone is irregular. The veinlets and gangue boundary rims of supergene chalcocite decrease in size and number with increasing depth until they practically disappear altogether. Below this zone supergene chalcocite is seen only in microscopic veinlets and accounts for only an insignificant proportion of the total copper. The chalcocite in this zone is not of the mottled type. Most of it is steely white like the deep-level chalcocite. Some of it, however, is distinctly bluish. This bluish chalcocite does not etch with dilute HNO_3 and in some respects behaves like covellite. In places this blue chalcocite is replaced by tiny veinlets of covellite. All blue chalcocite in Magma ores is found associated with the development of covellite or in supergene veinlets in bornite. This evidence indicates that this blue chalcocite is supergene. Similar blue chalcocite is found in the Kennecott mine, Alaska. It has been shown by Posnjak, Allen and Merwin that the blue color of the chalcocite was due to a small proportion of covellite contained in solid solution.[20] The presence of minute veinlets of covellite in blue chalcocite in Magma ores suggests that the blue color is due to oxidation of the chalcocite.

Covellite

There are two modes of occurrence of covellite in the Magma mine:

1. As a product of direct oxidation of chalcocite.
2. As a replacement of bornite, and more rarely of galena.

The first mode of occurrence is found only well up in the oxidation zone where the covellite is invariably associated with more or less malachite and mottled chalcocite. This is the only type of covellite which is observed in the hand specimen.

In the second mode of occurrence the covellite always occurs as veinlets or as rosettes of small plates along gangue boundaries. The total amount of covellite is insignificant in comparison with hypogene sulfides and does not exceed 2 per cent. in any polished section examined. On the 900-ft. level it is relatively abundant and from this level to the bottom of the mine is observed in a large number of the specimens examined, although only in very small amounts. The supergene derivation of this covellite is beyond question. From the 900-ft. level to the bottom there is a

[20] E. Posnjak, E. T. Allen, and H. E. Merwin: The Sulphides of Copper. *Econ. Geol.* (1915) **10**, 526.

progressive diminution in the proportion of covellite to bornite, and in sections from the 2000-ft. level the covellite is discernible only with higher magnification. The distribution of covellite is spotty; however, the tendency is in the same direction, as shown by the accompanying table, which shows the number of specimens containing covellite and the total number of specimens collected from each level.

DISTRIBUTION OF COVELLITE

Level	No. of Specimens Collected	No. of Specimens Containing Covellite	Percentage of Specimens Containing Covellite
900	9	7	78
1000	5	4	80
1100	8	1	12
1200	19	6	32
1300	4	1	25
1400	6	5	83
1500	4	1	25
1600	8	2	25
1700	8	4	50
1800	20	10	50
2000	21	4	19

An investigation was made to determine whether mine oxidation was responsible for the covellite. Specimens of bornite collected on the 1000-ft. level were found to carry no covellite, although the material came from a point just beneath a crust nearly a foot thick which was obviously the result of mine oxidation during the eight years that the drift had been open. If mine oxidation is competent to produce covellite, it should have been produced here. Another specimen which contained microscopic amounts of covellite was collected from the face of one of the crosscuts on the 1800-ft. level. The specimen was exposed to mine oxidation only a few hours before it was collected. Therefore this covellite can not be due to mine oxidation. This evidence is conclusive that no covellite is due to mine oxidation. From the above evidence, it is concluded that all the Magma covellite is supergene.

Although sphalerite enriches easily to covellite, no such enrichment was observed in polished sections of Magma ores. Although not abundant in the zone of enriched bornite ores, occasional areas of sphalerite are observed in polished sections of ores from this zone. The solutions were evidently sufficiently concentrated in copper to develop covellite as a replacement of bornite but not sufficiently concentrated to attack sphalerite.

Supergene Chalcopyrite

Some chalcopyrite occurs as veinlets and plates cutting bornite. Although not important quantitatively, it is widespread and persists to

the deepest workings of the mine. Some chalcopyrite occurs as veinlets and plates cutting bornite. Some of these veinlets and plates follow open spaces (cracks and seams) in the bornite. According to interpretations given earlier in this paper, these veinlets and plates are supergene. The replacement frequently takes place along parallel directions in the bornite, giving a series of parallel chalcopyrite plates. Frequently a second series of parallel chalcopyrite plates is formed at an angle to the first series. The effect of these two series of plates, as seen in polished section, is to form a grating.[21] The parallelism of these plates indicates that the replacement has followed definite crystallographic directions in the bornite. These gratings have been produced artificially by Zies, Allen and Merwin[22] both by immersing bornite in dilute sulfuric acid and by immersing bornite in a 5 per cent. solution of cupric sulfate. Such chalcopyrite gratings might therefore be expected to form at lower levels by the action on bornite of cupric sulfate and sulfuric acid which are produced in the zone of oxidation. On the other hand, chalcopyrite gratings in bornite indistinguishable from those just described have been produced artificially by Lombard and Merwin[23] by the action of heated sulfur vapor on bornite and by the unmixing of a bornite solid solution containing an excess of iron over that necessary to form pure bornite. The evidence is clear that chalcopyrite gratings in bornite could be produced by either hypogene or supergene agencies. If the chalcopyrite plates follow open spaces, the grating is of supergene origin, otherwise the structure does not give a definite clue as to its origin.

Deep-level Chalcocite

The term deep-level chalcocite is used to denote chalcocite which is intimately intergrown with bornite in graphics, mutual boundaries and some types of gratings.[24] Chalcocite in these structures most commonly occurs in the lower levels. Chalcocite-bornite gratings have not been observed in Magma ores.

It should not be inferred that all "deep-level" chalcocite is found in the lower levels; it has been observed as high as the 800-ft. level. Above this level a large proportion of the chalcocite is hypogene but the enrichment process was so powerful that the structures formed between the hypogene chalcocite and the bornite have been destroyed; thus the

[21] The term "grating structure" was recently proposed by L. C. Graton to supplant the term "lattice structure." The latter term is objectionable as it is apt to be confused with the molecular space lattice.

[22] E. G. Zies, E. T. Allen and H. E. Merwin: Some Reactions Involved in Secondary Copper Sulphide Enrichment. *Econ. Geol.* (1916) **11**, 476 and 479.

[23] H. E. Merwin: Oral communication.

[24] For discussion of hypogene and supergene gratings, see A. Locke, D. A. Hall and M. N. Short: Role of Secondary Enrichment in Genesis of Butte Chalcocite. A. I. M. E. *Trans.* (1924) **70**, 955; also Fig. 9.

principal criteria for the recognition of hypogene chalcocite are missing. "Deep-level" chalcocite is chalcocite intergrown with bornite in structures indicative of contemporaneity.

The graphic structure needs no introduction. It has been described in ores from many localities. True graphic structures are rather rare in the Magma mine. Patterns closely resembling graphics but lacking their regularity are more common. The discussion of graphics applies equally well to subgraphics. Although graphic chalcocite-bornite structures are common, most of the deep-level chalcocite occurs as irregular rounded blebs in bornite. The two minerals have mutual boundaries. Other patterns formed by bornite and chalcocite show sharp-pointed projections of chalcocite into bornite, suggesting slight replacement of bornite by the chalcocite.

Another mineral with precisely the same type of outlines is closely associated with deep-level chalcocite. It is about the same hardness as chalcocite but has a pinkish-yellow tinge and is only slowly affected by nitric acid. Repeated efforts were made to identify this mineral but it occurs in such small quantities that it could not be isolated. Its properties and associations are identical with those of a mineral occurring in Butte ores which, according to Rogers, was identified by Laney as klaprotholite ($Cu_6Bi_4S_9$).[25] We have not confirmed the identification of this mineral and refer to it as the "pink unknown mineral." Intergrowths between this pink unknown mineral and bornite are exactly like those between bornite and chalcocite (Fig. 27). Klaprotholite has never been observed in veinlets following open spaces. Rogers states that in Butte klaprotholite is hypogene. The chemical similarity of klaprotholite to tetrahedrite makes it probable that it is hypogene, since tetrahedrite is nearly always if not always hypogene. If deep-level chalcocite were supergene, then from the similarity of structures it would follow that the pink unknown mineral, if klaprotholite, would likewise be supergene.

The distribution of deep-level chalcocite is very "spotty." It seems strange that very little was found above the 1400-ft. level. Below that level it is of rather common occurrence, but even where it does occur its distribution is very irregular. Deep-level chalcocite and covellite rarely occur together. The reason for this lack of association is not known. Frequently, however, in the same crosscut one specimen contains chalcocite and another not far from it, covellite. Assuming that deep-level chalcocite is hypogene, it is reasonable to expect that it would be modified by enriching solutions in the upper levels. On the other hand, much chalcocite of undoubted supergene origin is of the steely white variety and is indistinguishable physically and chemically from deep-level chalcocite.

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[25] A. F. Rogers: The So-called Graphic Intergrowth of Bornite and Chalcocite. *Econ. Geol.* (1916) **11**, 585.

Assuming that a graphic intergrowth of bornite and chalcocite is acted upon by enriching solutions, the bornite would be replaced in part by supergene chalcocite and this chalcocite would be indistinguishable from the original hypogene chalcocite. It would be expected that the supergene chalcocite molecules would orient themselves with the original chalcocite molecules much as solution quartz in quartzite orients itself with the original quartzite grains. The graphic patterns would thus be obliterated. Hence the failure to find these graphic structures in the upper levels is not proof that they did not exist.

In general deep-level chalcocite is found only with richer bornite ores, but not all rich bornite ores contain chalcocite. The accompanying table shows the ratio of the specimens containing deep-level chalcocite to the total number of specimens collected.

Distribution of Chalcocite

Level	No. of Specimens Collected	No. of Specimens with Chalcocite	Percentage of Specimens Containing Chalcocite to Total
900	9	1	11
1000	5	1	20
1100	8	0	0
1200	19	1	5
1300	4	0	0
1400	6	1	17
1500	4	1	25
1600	8	0	0
1700	8	5	62
1800	20	2	10
2000	21	7	33

All of these specimens contain ore minerals but not all of them contain bornite. Between the 1600-ft. and 2000-ft. levels this table furnishes a fairly reliable estimate of the distribution of chalcocite. Between these levels all of the crosscuts were available for sampling, very little ore having been mined out, and the sampling was carried out in a uniform manner. From the 1200-ft. to the 1600-ft. levels, the suite collected was not representative, as the richest ores had been removed. From the 500-ft. to the 1200-ft. levels collections were gathered by different investigators on three different occasions, and consequently are fairly representative of these levels. From the 800-ft. to the 1000-ft. levels the orebody approaches within a few feet of No. 2 shaft and a pillar of ore is left for shaft protection. The sampling from this pillar was adequate.

The distribution of deep-level chalcocite is thus very spotty, but according to the table the proportion of chalcocite to bornite remains approximately constant with depth. The actual quantity of the bornite, and with it the quantity of chalcocite, increases with depth.

Summary of Mineralization and Enrichment

The hypogene ore minerals of the Magma mine are bornite, chalcopyrite, chalcocite, tennantite, sphalerite, and galena. Pyrite is locally very important but in some of the richer bornite shoots it is of subordinate importance. The ores were enriched in the period between their deposition and the tilting of the sediments and diabase in which the orebodies occur. Following the tilting, a great thickness of dacite was poured out covering the outcrop of the vein to a depth of many hundreds of feet. A period of faulting came next, probably soon after the dacite extrusion. The scarp on the eastern or upthrown segment was eroded back, revealing a part of the outcrop of the vein. This erosion is believed to be of relatively recent geological age, probably Quaternary. Since the outcrop was laid bare, there has been a second period of oxidation and enrichment superimposed on the earlier period. It did not extend below the 500-ft. level and is of subordinate importance.

The hypogene ores were rich, and pyrite could not have been abundant near the apex of the oreshoot, as there are no limonite masses, such as might be expected from an orebody as large as this one. The surface at the time of the first oxidation must have been relatively flat, and it is hard to believe that iron sulfate solutions were carried off before they had an opportunity to form limonite. The oxidation was not carried to completion, and massive chalcocite and covellite ores are mixed with oxides and carbonates. The depth to which oxidation extended and its incompleteness indicate arid conditions and it is probable that a considerable part of the enrichment took place above the water level. The bottom of oxidation, however, corresponded very closely to the pre-tilting water level. The zone of supergene sulfides extends a relatively short distance, 100 to 150 ft. below the bottom of oxidation, and is likewise tilted. Below this zone, enrichment is wanting altogether or is represented by tiny veinlets of covellite or chalcocite.

In order to explain the total amount of supergene sulfide ores, it is not necessary to assume the oxidation and removal of a great depth of hypogene ores, especially as the hypogene ores themselves were rich. It is probable that the present surface is not far from the apex of the Magma oreshoot when it was formed and that near this apex the stoping length of the orebody was small in comparison with the stoping length at greater depths. This would account for the relatively shallow depth of enrichment.

From the data given it is seen that the amount of deep-level chalcocite in the lower levels of the mine forms only a small proportion of the total ore; not over 10 per cent. and probably much less. It is probable that the amount of chalcocite present at the greatest depths mined is not in excess of that which could theoretically be furnished by the oxidation and

solution of hypogene ores above the water level and precipitation of the copper in these solutions at lower levels. Nevertheless the distribution of deep-level chalcocite is not what one would expect were it supergene. It does not diminish from the water level downward but actually increases. Tiny veinlets of undoubted supergene chalcocite are encountered in the lower levels and these gradually diminish in size and quantity from the water level downward, as one would expect. Covellite behaves in the traditional supergene way. Veinlets of covellite and supergene chalcocite cut the deep-level chalcocite (Fig. 22), hence both covellite and supergene chalcocite are distinctly later than the deep-level chalcocite. From the facts stated above, we are led to believe that the deep-level chalcocite of the Magma mine is hypogene.

Note

The photomicrographs have been selected to give an outline of the story of ore deposition and enrichment at the Magma mine which can be understood almost without reference to the text. They are arranged in sequence so as to depict polished sections of samples taken from the top of the zone of oxidation down through the successive ore zones to the bottom of the mine. (The fact that the successive polished sections are not always from samples taken at successively lower elevations in feet does not render necessary modification of this statement.) The following abbreviations are used on the photomicrographs: cc, chalcocite; cv, covellite; bn, bornite; cp, chalcopyrite; sl, sphalerite; gn, galena; tn, tennantite; py, pyrite; ag, native silver; im, limonite; qtz, quartz.

FIG. 12.—PYRITE OXIDIZING TO LIMONITE IN PRESENCE OF COVELLITE. PSEUDOMORPHS OF LIMONITE ARE SHARP AND CLEAN-CUT. AREA BETWEEN PSEUDOMORPHS IS PRACTICALLY ALL COVELLITE WHICH HAS REPLACED CHALCOCITE, ONLY A FEW SMALL TRIANGULAR PATCHES OF CHALCOCITE REMAINING UNALTERED. DIFFERENT SHADES OF COVELLITE ARE DUE TO ITS ANISOTROPISM. PROCESS IS INDICATIVE OF STRONGLY OXIDIZING CONDITIONS. PSEUDOMORPHS OF LIMONITE AFTER PYRITE ARE PROBABLY DUE TO NEUTRALIZATION BY CALCIUM CARBONATE SOLUTIONS OF SULFURIC ACID GENERATED IN OXIDATION OF PYRITE. 500-FT. LEVEL 1-W RAISE, 30 FT. BELOW 400-FT. LEVEL. × 100.

FIG. 13.—COVELLITE NEEDLES REPLACING CHALCOCITE. THIS STRUCTURE IS FOUND ONLY HIGH IN SUPERGENE CHALCOCITE ZONE. IT IS DUE TO SUPERIMPOSITION OF DIRECT OXIDATION ON SUPERGENE ENRICHMENT. MALACHITE IS PRACTICALLY ALWAYS FOUND ACCOMPANYING COVELLITE. CHALCOLITE IS CRUMBLY AND FULL OF PITS, AND PROCESS OF OXIDATION IS ACTUALLY A COPPER IMPOVERISHMENT. 500-FT. LEVEL, 1½ RAISE. × 100.

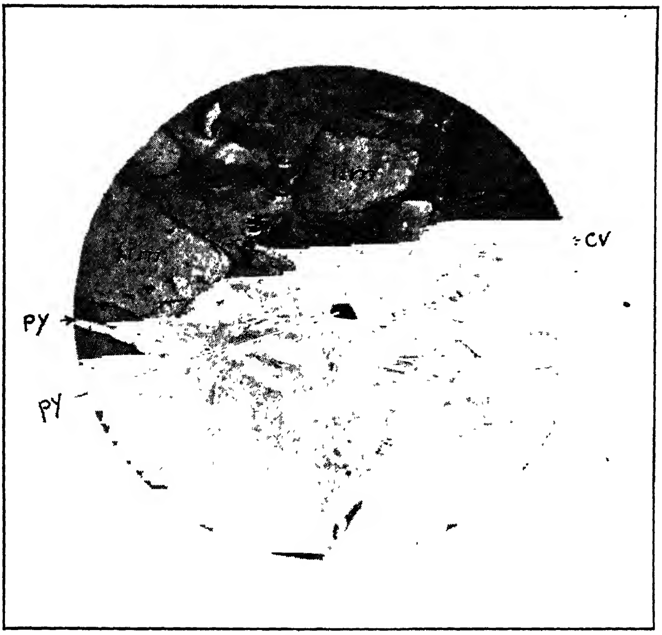


FIG. 12.

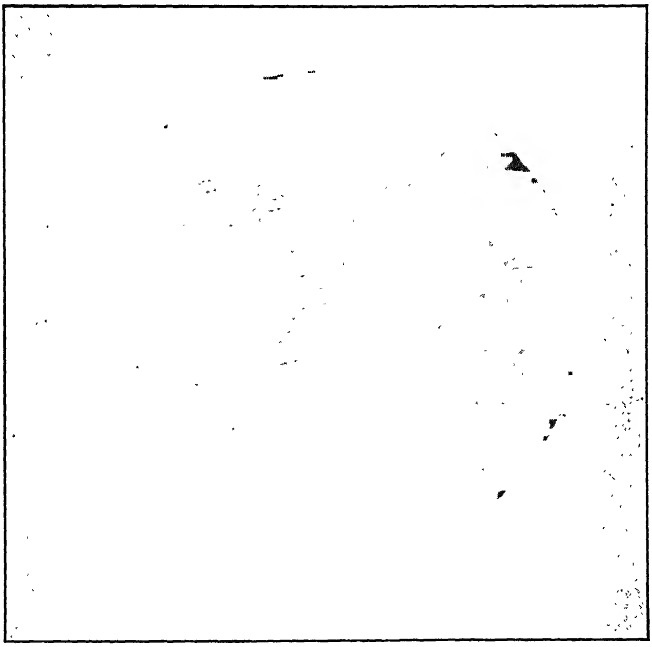


FIG. 13.

Fig. 14.—Chalcocite replacing pyrite. Enrichment process is very intense here and no hypogene copper sulfides remain unreplaced. Outline of a large pyrite grain in midst of chalcocite area in upper left quadrant of picture is very clear, but only a few small patches of pyrite along margins of former grain have escaped replacement by chalcocite; interior of grain is now all chalcocite. Strongly indicative of supergene replacement. 500-ft. level, W 1½ stope. × 100.

Fig. 15.—Chalcocite replacing pyrite. Replacement has followed crystallographic directions (probably octahedral) in pyrite. This type of replacement, although not common, is occasionally encountered in upper part of supergene chalcocite zone. It has not been observed in deep-level chalcocite. 650 east drift at 7 raise. × 290.

Fig. 16.—Supergene chalcocite (light) replacing bornite (dark). Larger areas of bornite are traversed by tiny cracks of chalcocite. By progressive widening of these cracks and rounding of sharp edges of bornite areas, unreplaced bornite is left as formless masses surrounded by chalcocite. This type of replacement has been called by Professor Tolman the ice-cake structure. It is found only in enrichment zone and has not been observed in deep level chalcocite. 900-ft. level, E 2½ stope, 20 ft. below 800-ft. level. × 100.

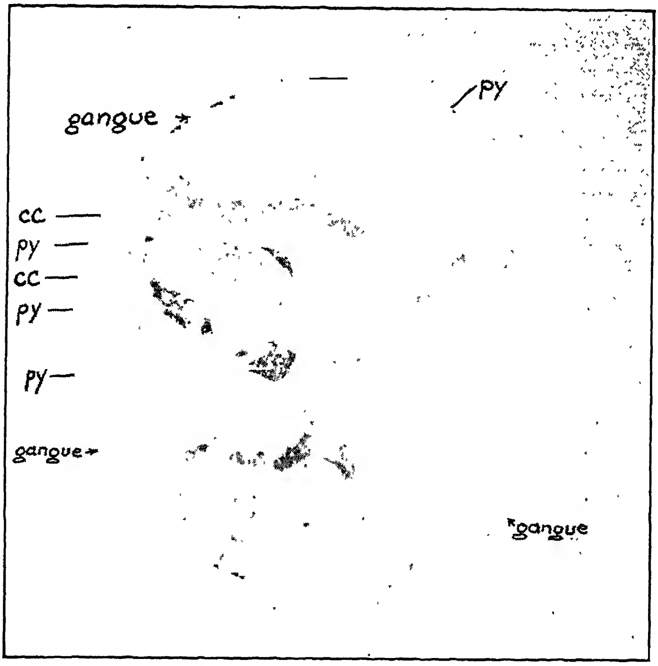


Fig. 14.

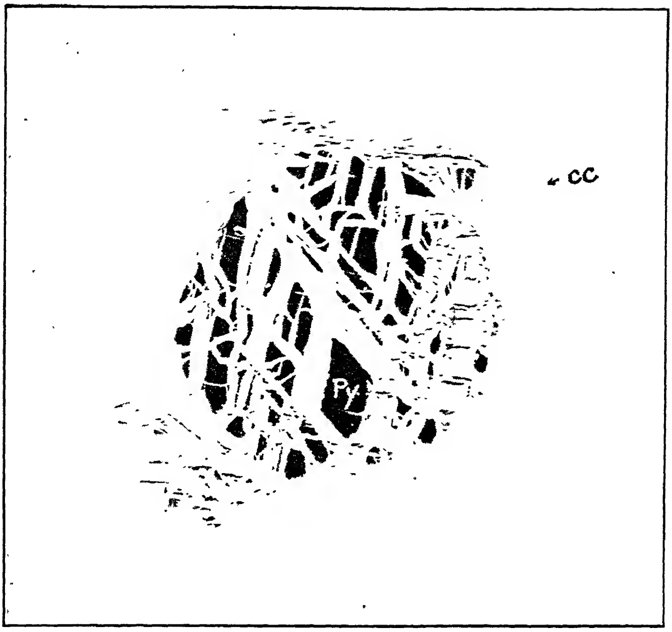


FIG. 15.

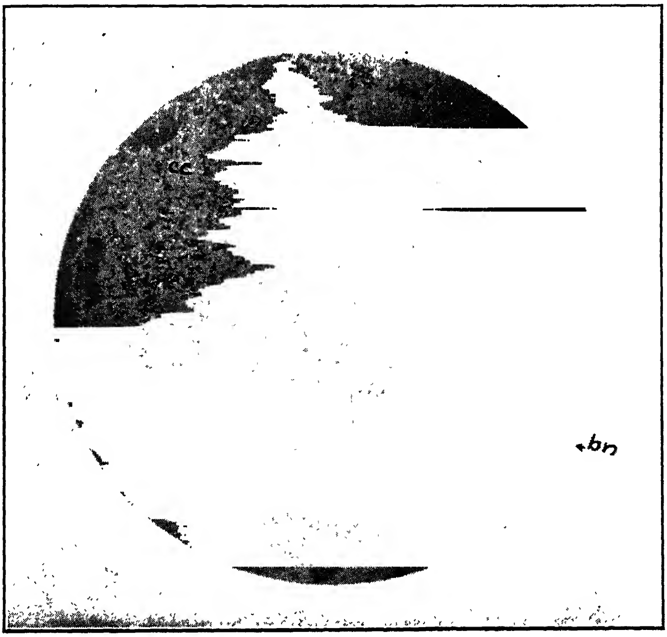


FIG. 16.

FIG. 17.—COVELLITE REPLACING BORNITE ALONG CRYSTALLOGRAPHIC DIRECTIONS. HABIT OF COVELLITE IS DIFFERENT FROM THAT SHOWN IN FIGS. 12 AND 13. INSTEAD OF ELONGATED PLATES IRREGULARLY DISTRIBUTED, COVELLITE APPEARS AS CLUSTERS OF SMALL PLATES IN PART FOLLOWING ACTUAL OPEN SPACES IN BORNITE. COVELLITE IS DUE TO SUPERGENE REPLACEMENT BY DOWNWARD-MOVING SOLUTIONS AND NOT TO DIRECT OXIDATION. 900-FT. LEVEL 25 FT. EAST OF 1-W RAISE. × 100.

FIG. 18.—COVELLITE REPLACING BLUE CHALCOCITE ALONG SHRINKAGE CRACKS. COVELLITE ALSO FORMS ALONG GANGUE VEINLET IN CHALCOCITE. BORNITE HAS NO COVELLITE. CHALCOCITE AND BORNITE SHOW MUTUAL BOUNDARY PATTERNS AND BOTH ARE HYPOGENE. (PHOTOGRAPHED THROUGH BLUE COLOR SCREEN, BRINGING OUT CONTRAST BETWEEN BORNITE AND BLUE CHALCOCITE BUT NOT GOOD CONTRAST BETWEEN BLUE CHALCOCITE AND COVELLITE.)

FIG. 19.—SAME VIEW AS FIG. 18, BUT PHOTOGRAPHED THROUGH GREEN COLOR SCREEN. SHOWS GOOD CONTRAST BETWEEN COVELLITE AND BLUE CHALCOCITE BUT POOR CONTRAST BETWEEN BLUE CHALCOCITE AND BORNITE. 900-FT. LEVEL EAST OREBODY. × 290.

FIG. 17.

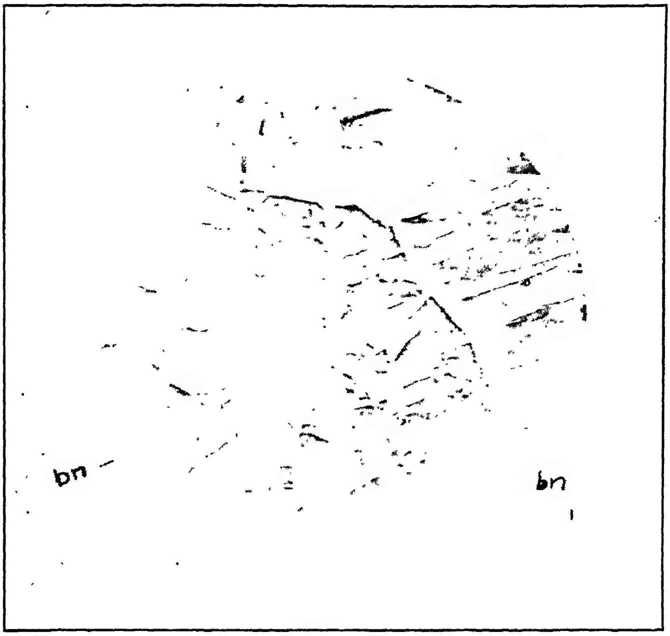


Fig. 18.

Fig. 19.

FIG. 20.—BORNITE AND BLUE CHALCOCITE IN SUBGRAPHIC STRUCTURE. PHOTOGRAPHED THROUGH BLUE COLOR SCREEN. FROM SAME SECTION AS FIG. 18.

FIG. 21.—SAME VIEW AS FIG. 20. PHOTOGRAPHED WITH GREEN COLOR SCREEN. CHALCOCITE AREAS SHOW TWO VARIETIES. LIGHTEST COLORED AREAS ARE WHITE CHALCOCITE WHICH IS OF HYPOGENE ORIGIN; PARTLY ALTERED BY SUPERGENE PROCESSES TO BLUE CHALCOCITE, *i. e.*, CONTAINING COVELLITE IN SOLID SOLUTION. PICTURE DOES NOT INDICATE SUPERGENE DERIVATION OF BLUE CHALCOCITE. AS EXPLAINED IN TEXT, SUPERGENE ORIGIN OF BLUE CHALCOCITE IS INFERRED FROM ITS DIMINUTION IN DEPTH. ALONG OPEN SPACES WHERE SOLUTIONS MOVED MOST FREELY, PROCESS WAS CARRIED ONE STEP FARTHER, RESULTING IN REPLACEMENT OF CHALCOCITE BY COVELLITE. BORNITE IS APPARENTLY UNCHANGED. SOLUTIONS WERE PROBABLY POOR IN COPPER BUT RICH IN FERRIC SULFATE. PROCESS REPRESENTS BEGINNING OF AN IMPOVERISHMENT OF ORE BY REMOVAL OF EASILY REMOVED ATOM OF COPPER FROM Cu_2S. $\times$ 290.

FIG. 22.—CHALCOCITE IN VEINLETS CUTTING CHALCOCITE AND BORNITE INTERGROWTH. CHALCOCITE IN LARGER AREAS IS OF DEEP-LEVEL TYPE. IT SHOWS MUTUAL BOUNDARY AND SUBGRAPHIC PATTERNS WITH BORNITE. CHALCOCITE IN VEINLETS IS DISTINCTLY LATER. IT FOLLOWS CRACKS IN BORNITE. WHERE VEINLETS PASS INTO LARGER CHALCOCITE AREAS, CHALCOCITE ALMOST LOSES ITS IDENTITY BUT CONTINUITY OF VEINLET IS SHOWN BY SCATTERED CRACKS. CRACKS ARE LATER THAN DEEP-LEVEL CHALCOCITE, AND CHALCOCITE IN VEINLETS IS LATER THAN CRACKS WHICH IT FOLLOWS; HENCE TWO GENERATIONS OF CHALCOCITE ARE REPRESENTED. TO ASSUME THAT DEEP-LEVEL CHALCOCITE IS SUPERGENE WOULD BE TO POSTULATE TWO DISTINCT PERIODS OF ENRICHMENT. IT IS BELIEVED THAT DEEP-LEVEL CHALCOCITE IS HYPOGENE AND CHALCOCITE IN VEINLETS IS SUPERGENE. 2000-FT. LEVEL, WEST DRIFT. $\times$ 100.

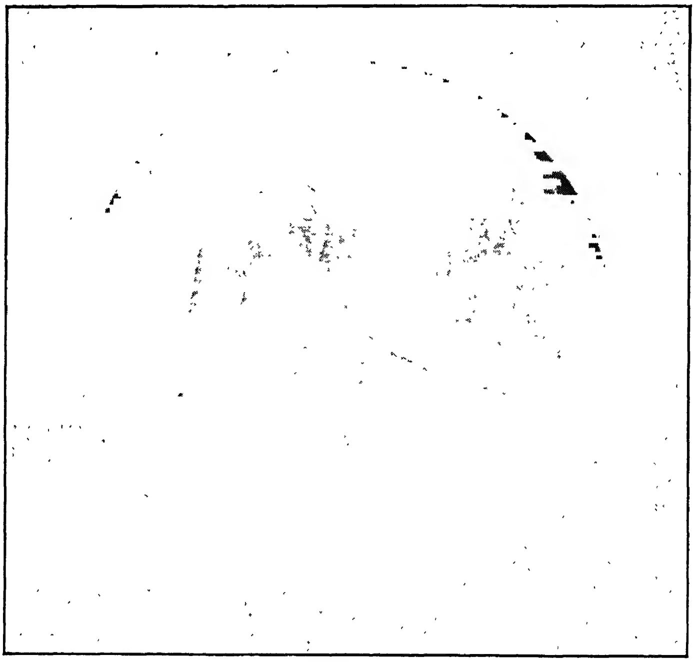


FIG. 20.

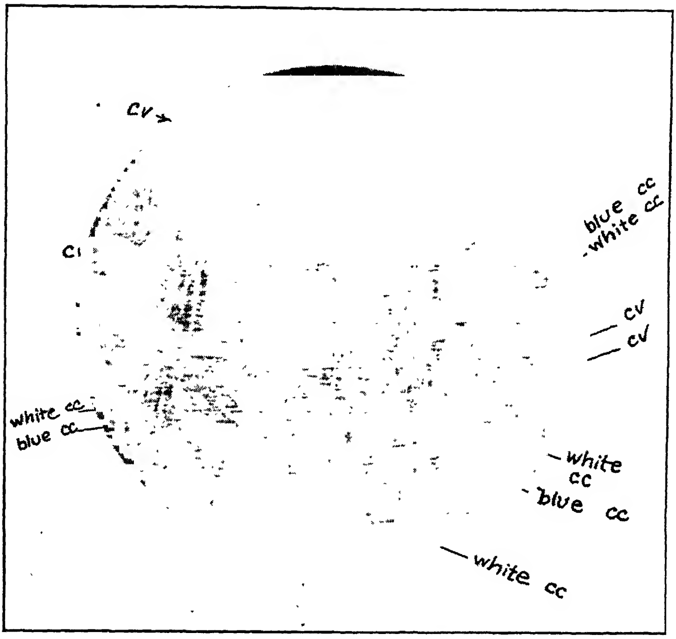
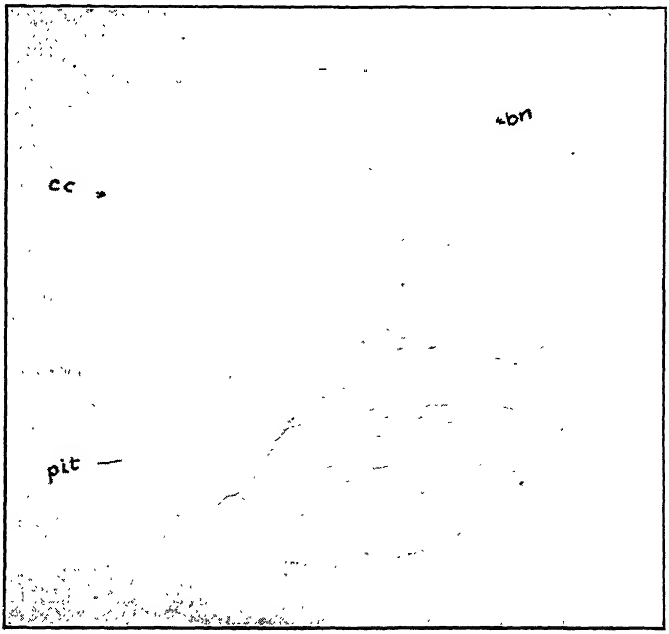

cv
blue cc
white cc
c
cv
cv
white cc
blue cc
white
cc
blue cc
white cc

FIG. 21.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cc
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FIG. 22.

FIG. 23.—DEEP-LEVEL CHALCOCITE (LIGHT) AND BORNITE (DARK). CIGAR-SHAPED BORNITE AREAS SHOW PRONOUNCED PARALLELISM. BETWEEN BORNITE "CIGARS" IS A SUBGRAPHIC INTERGROWTH OF CHALCOCITE AND BORNITE. IT IS BELIEVED "CIGARS" ARE SKELETAL CRYSTALS OF BORNITE WHICH WERE FIRST TO FORM. WITH A DECREASE IN IRON CONTENT, REMAINING MATERIALS IN SOLUTION SEPARATED OUT AS SIMULTANEOUS DEPOSITION OF CHALCOCITE AND BORNITE. 2000-FT. LEVEL, 15 CROSSCUT. × 100.

FIG. 24.—FROM SAME SECTION AS FIG. 23. SOME OF LARGER BORNITE AREAS ARE CROSSED BY IRREGULAR VEINLETS OF CHALCOCITE. WITH FURTHER DECREASE IN IRON CONTENT, CONDITIONS WERE SUCH THAT ONLY CHALCOCITE DEPOSITED. SOLUTIONS DEPOSITING CHALCOCITE HAD SOME REPLACING ACTION ON BORNITE WHICH TOOK FORM OF VEINLETS. THESE CHALCOCITE VEINLETS LACK CONTINUITY AND REGULARITY OF SUPERGENE CHALCOCITE VEINLETS AND DO NOT FOLLOW CRACKS AND GANGUE BOUNDARIES. ALL THE CHALCOCITE IS BELIEVED TO BE HYPOGENE. × 290.

FIG. 25.—GRAPHIC CHALCOCITE AREAS IN BORNITE. TENNANTITE AREA AT LEFT OF FIELD SHOWS MUTUAL BOUNDARIES WITH BORNITE. 2000-FT. LEVEL, WEST DRIFT, 8 CROSSCUT AT FOOT WALL. × 100.

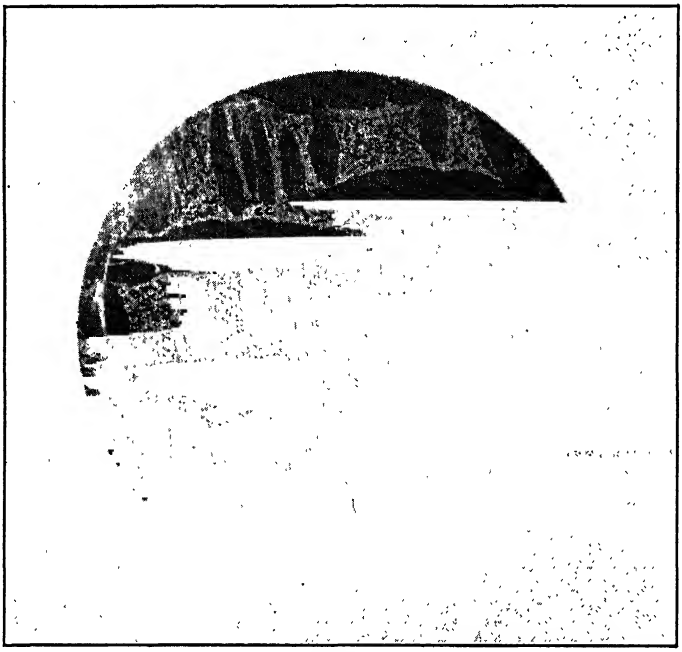

FIG. 23.

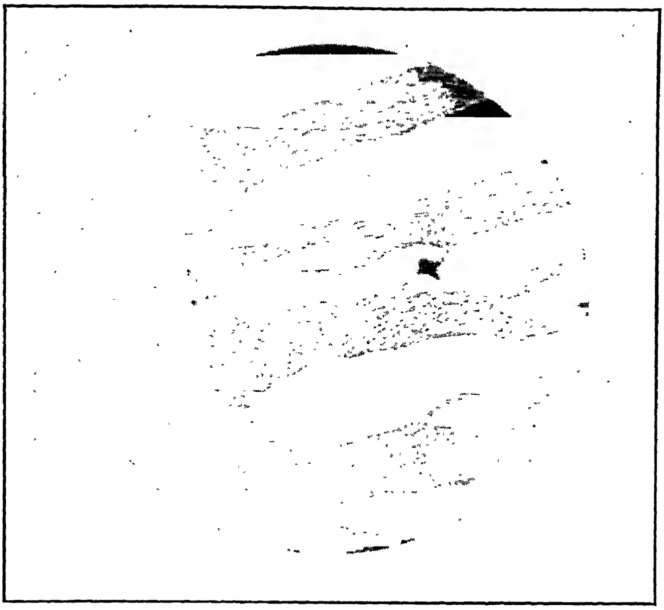

FIG. 24.

FIG. 25.

FIG. 26.—CHALCOPYRITE AND BORNITE IN MUTUAL BOUNDARY PATTERNS. THEIR PERSISTENCE IN DEPTH PROVES THAT BOTH MINERALS ARE HYPOGENE. THIS IS A CONFIRMATION OF ARGUMENT THAT MUTUAL BOUNDARY STRUCTURES ARE PROOF OF HYPOGENE ORIGIN OF MINERALS IN SUCH RELATIONS. OBSERVE SIMILARITY TO CHALCOCITE-BORNITE PATTERNS IN FIG. 22. 2000-FT. LEVEL, WEST DRIFT, 11 CROSSCUT. × 100.

FIG. 27.—PINK UNKNOWN MINERAL AND CHALCOCITE INTERGROWN WITH BORNITE IN SUBGRAPHIC STRUCTURES. PINK UNKNOWN MINERAL HAS EXACTLY SAME STRUCTURE AS DEEP-LEVEL CHALCOCITE AND IS INVARIABLY ASSOCIATED WITH IT. THEY MUST, THEREFORE, HAVE SAME ORIGIN. 1400-FT. LEVEL, WEST DRIFT, 6½ STOPE. × 100.

FIG. 28.—GRAPHIC PATTERN OF GALENA IN BORNITE. GALENA IS INVARIABLY HYPOGENE. GRAPHIC STRUCTURE IS, IN THIS INSTANCE, A PROVED HYPOGENE STRUCTURE. CONCLUSION IS THAT CHALCOCITE IN SIMILAR STRUCTURAL RELATION TO BORNITE IS LIKEWISE HYPOGENE. 1000-FT. LEVEL, EAST DRIFT, EAST OREBODY. × 290.

FIG. 26.

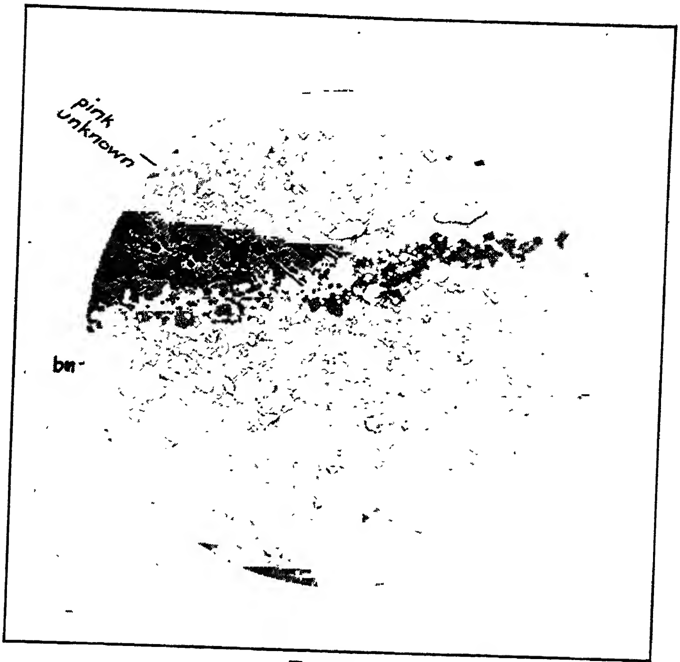

FIG. 27.

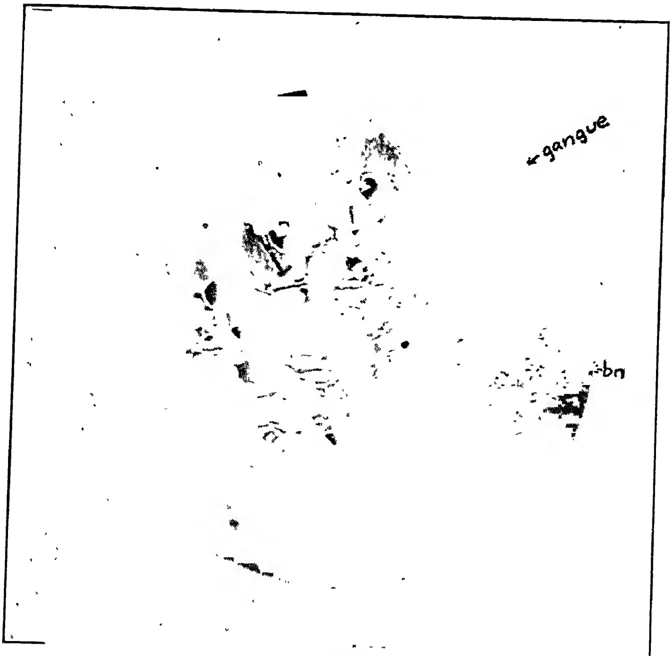

FIG. 28.

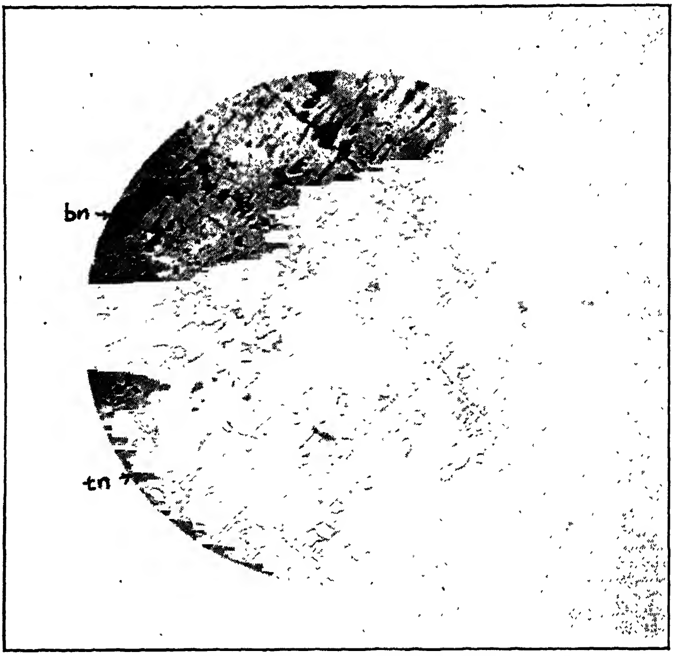

FIG. 29.—TENNANTITE (LIGHT) AND BORNITE (DARK) IN PATTERN CLOSELY RELATED TO STANDARD MUTUAL BOUNDARY PATTERN. ALTHOUGH SOME PROJECTIONS OF BORNITE INTO TENNANTITE ARE SHARPER THAN ANY PROJECTIONS OF TENNANTITE INTO BORNITE, OVER MUCH GREATER PART OF THEIR LINEAR EXTENT BOUNDARIES BETWEEN THE TWO MINERALS HAVE TRUE MUTUAL CHARACTER. BOTH MINERALS CONTINUE IN UNDIMINISHED QUANTITIES TO DEEPEST LEVELS AND FOR THIS REASON ARE HELD TO BE HYPOGENE. A PATTERN CLOSELY RELATED TO MUTUAL BOUNDARY PATTERN IS HERE PROVED TO BE PRODUCT OF HYPOGENE DEPOSITION. 2000-FT. LEVEL, WEST DRIFT, 17 CROSSCUT. $\times$ 100.

DISCUSSION

L. C. GRATON, Cambridge, Mass.—I do not believe that deep chalcocite is hypogene, but as I have been unable to prove it to men who have worked side by side with me, month after month and year after year, the proof cannot be very evident.

MEMBER.—There are certain generalizations which have developed in my mind over a long period of years during which I have been more or less familiar with this property, and I am convinced that he is right in that it is hypogene chalcocite.

W. LINDGREN, Cambridge, Mass.—I do not believe that the interlocking boundaries between chalcocite and bornite indicate simultaneously deposition.

Geology of the Yoquivo, Chihuahua, Mining District

By C. W. Hall,* Trenton, N. J.

(New York Meeting, February, 1926)

Owing to its isolation and comparatively small tonnage, the Yoquivo district is not widely known; though financially important and, geologically, quite interesting.

San Francisco de Yoquivo, the center of the mining district, is in the western part of the state of Chihuahua, Mexico; about 35 miles west of San Juanito, on the Kansas City, Mexico & Orient R. R. The town can be reached from the railroad by a good trail over a rugged country.

The elevation of Yoquivo is 6500 ft., and peaks surrounding the town are 2000 ft. higher. The mining towns of Concheño and Ocampo are 25 miles northwest, and Uruachic is 20 miles southwest. The climate, temperate and healthful, is almost ideal. The rainy season begins in July and ends the latter part of September. There is usually, though not always, some precipitation in all months except April, May, and June. The mountains are covered with pine and scrub oak. Water, wood and lumber are plentiful for all ordinary purposes, and hydroelectric power is available for about six months of the year, when wood for steam power is utilized as an expedient. The district has been worked recurrently for over 60 years.

The Yoquivo province lies on the steep western slope of the Tarahumara division of the Sierra Madre range of mountains. The province is traversed by many deep canyons, most of which flow west. The town is situated on the San Francisco arroyo, which makes a confluence with the Huevachic stream 2½ m. southwest; the Huevachic in turn joins the Candemeña river farther down and these two drain the western flank of the Tarahumara mountains and form the headwaters of the Mayo river.

The claims of the Yoquivo Development Co., which is the only organization in the district with a record of any considerable production, lie within an area roughly 1½ m. square. The area extends from the San Francisco arroyo north, and from the Dolores arroyo west, and is confined to the andesite formation.

General Geology

The terrane about Yoquivo is composed of a series of Tertiary eruptives that consist of numerous phases of andesite and rhyolite, and

* Vice president, in charge of production, United Clay Mines Corp.

aggregate several thousand feet in thickness. Volcanic rocks only are visible for a radius of miles; erosion and trenching by the deepest canyons have failed to disclose any definite basal formation. Sediments known to exist at some distance from the district indicate that the volcanics may rest upon those strata. A few fragments of granite float have been discovered, but the source is not apparent. It possibly may be related to the deep-seated magma from which the rhyolite was evolved.

The main exposure of the vein-bearing quarter is largely due to an uplift accompanied by faulting. The andesite area is extensive and is older in age than the rhyolite capping of 2,500 ft. minimum thickness. The largest and most prominent dislocation of the district is called No. 1 fault. The displacement has left the rhyolite and andesite in juxtaposition along a well-defined line of contact, easily recognized because of the light-colored rhyolite on one side, and the dark andesite on the other.

Descriptive Geology

The rhyolites are a succession of flows and fragmental deposits, some of which were interrupted by unconformities of erosion. The larger part is tuff; much is well stratified and denotes deposition in water; the color is red and white; alteration is noticeable and many beds have been highly indurated by silicification. The upper flow is conspicuous by reason of its reddish color and tendency to weather in bold cliffs.

The rhyolites afford little evidence of having been invaded by mineralizing solutions and, although the andesite may not have been mineralized before the formation of some of the rhyolite series, the last extrusion of rhyolite probably came after the vein formation.

As the andesite is the vein-bearing formation of the district, geological study has been confined chiefly to that division, and the descriptive details herein outlined are limited to the territory circumjacent to the San Francisco mine.

The area is composed of severai andesitic phases, including flows, tuffs, dikes, sills, stocks, and likely a more deep-seated laccolith, to which is ascribed the general uplift. The andesite column has been disturbed by later faults and intrusions, and the geological relations of the section have not been satisfactorily solved. At one time a thorough investigation of the structural geology of the area was contemplated, but it was decided that the importance of the results would not justify the work involved.

A large part of the andesitic area is composed of tuffs; much is well stratified and is, in places, so thinly bedded as at a glance to resemble shale. Most of the beds have been tilted by intrusions and faults. The tuffs differ in texture and are both fine and coarse grained. The color is green, light green, and in places nearly white. Alteration is common in some beds in the form of silicification. The cementing material is largely

secondary calcite. The fragments consist of feldspar, chloritic material and calcite. A few veins of the district are in tuff, but the main orebodies in the San Francisco mine are in the massive andesite.

The massive andesite constitutes a series of flows and intrusives. The color ranges from black to light green, or even white, where alteration is severe along veins. The formation is characterized by weathering in sharp pinnacles. Although contact-metamorphism is not usually widespread, some contacts show positive zones of alteration; the intrusive itself is commonly altered as much, and in places more than the intruded rock. Phases have been classified as hornblende- trachy-, and augite-andesite, and, in addition, feldspar-porphyry, and latite.

The hornblende-andesite, which is next to the andesite-tuff in extent, is medium soft, and ranges from dark green to reddish purple. The texture is porphyritic and the ground-mass contains phenocrysts of plagioclase, which represents about 80 per cent. of the mineralogical composition. Hornblende and pyrite are present in variable amounts. The hornblende has been largely altered to chlorite and the pyrite is altered to hematite. Secondary calcite is usually abundant, and magnetite occurs in small proportions. Where the rock is stained by secondary hematite it assumes a reddish or purplish hue. The hornblende-andesite exists only as a flow. The orebodies discovered in the south part of the San Francisco mine—as well as many veins of the district—are for the most part in this formation.

The trachy-andesite is a lighter and more acid phase of the parent magma. The prevailing color is light green, but by kaolinization it becomes almost white. The ground-mass is fine and phenocrysts are small. Plagioclase is predominant, but orthoclase is also present in small amounts. The ferromagnesian minerals are chlorite and epidote. Magnetite is absent. Kaolin is abundant, and adularia occurs sparingly at points in close proximity to veins where low-temperature hydrothermal activity is manifest. The trachy-andesite is a flow. Valuable orebodies have been found in this formation.

The augite-andesite is the most basic rock of the group. Unaltered specimens are hard and brittle, black in color, and coarsely crystalline. Lath-shaped phenocrysts of labradorite are plentiful; labradorite constitutes about 60 and augite about 20 per cent. of the mineralogical composition. In places magnetite is present in sufficient quantities to deflect the magnetic needle. Along veins or margins of intrusions where metamorphosis has been severe, the augite-andesite is altered to a soft, white rock that is difficult to classify. The formation occurs both as an intrusive and as a flow. The veins in the north part of the San Francisco mine are in an intrusion of augite-andesite on 7 level. At other places in the mine the rock is noted as narrow dikes or tongues ramifying from the main body.

The feldspar-porphyry is probably an intermediate issue from the main magma. The color is grayish-green and dark red. The texture is porphyritic, and the rock contains well-defined phenocrysts of feldspar. Orthoclase and oligoclase are present in nearly equal amounts, and primary quartz sparingly. Kaolinization has not been severe. Specimens from along veins bear evidence of low-temperature hydrothermal alteration, and adularia is present as a result. The porphyry, which is found contiguous to valuable veins, is considered intrusive, and the mode of occurrence imports a significant relationship with the genesis of the principal orebodies.

The latite is likely a phase of the porphyry. Indeed, the two are wellnigh indistinguishable, and association with orebodies is identical.

Structural Geology

Perplexing faults are numerous throughout the andesitic area. All are probably normal; that is, the hanging wall is the downthrown side. With few exceptions little is known as to actual displacements.

Faults played a capital rôle in the Yoquivo mineral district; first, they afforded channels for the entry of original mineralizing solutions; second, they assisted in the exposure of the mineral-bearing andesitic area, and third, they interposed complexities by post-mineral displacements. The veins have been formed in fault fissures, the most of which strike N. 5°–40° E., and dip from 60°–75° eastward. The foremost veins exhibit strong evidence of slipping after the ore deposition was complete.

The five chief faults—all but one traversing the vein system—are numbered in order of their intersection on the adit level of the San Francisco mine, as follows: Nos. 1, 2, 3, 4, and 5 (see Fig. 1). Other displacements are to be noted, but most of them have had to do with the initial uplift and are extrinsic to recognized orebodies.

No. 1 fault strikes N. 57° W., and dips 75° to the west. This is the main rhyolite-andesite displacement that assisted materially in laying open the andesitic area, with consequent exposure of the entire mineralization of the Yoquivo district; that is, after the upthrow of the foot wall side, the thick capping of rhyolite was eroded away which unmasked the vein-bearing andesite beneath. No. 1 fault bounds the mineral area on the south. The break is probably post-mineral—at least later than the first stages of deposition. The amount of displacement is not known, but it is probably at least 1000 ft., and may be much more.

No. 2 fault strikes N. 40° E. and dips 65° to the east. At one time this fault was considered of signal importance, as it had apparently dislocated the main vein on the northerly trend. After an ample amount of development work, based on this premise, it was concluded that No. 2 fault was the result of late movement along the vein; the slipping, which took place along the foot wall, was deflected to the north at a slight angle

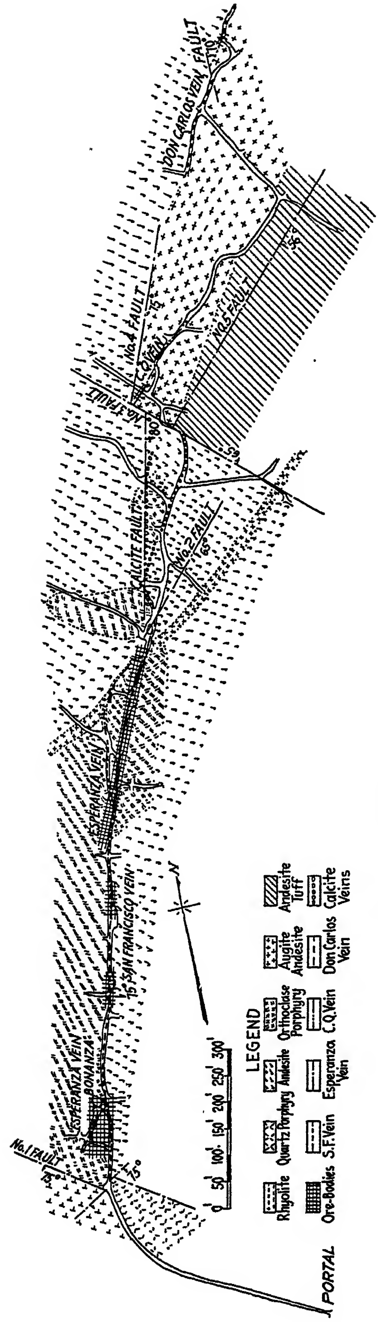

FIG. 1.—PLAN OF LEVEL 7, SAN FRANCISCO MINE.

to the strike, so that a shear zone was produced. All orebodies terminated abruptly at the so-called fault and solution of the problem was, for a time indeed, baffling.

No. 3 fault strikes N. 55° W. and dips 65° to the northeast. The total throw is not known. The fault is of interest in that the displacement dropped the north section of the andesite quarter down a sufficient distance to have afforded the veins protection in that area against complete erosion of their upper barren extremities; thus leaving a trivial showing at the outcrop. After its formation No. 5 fault was mineralized by barren quartz, accompanied by silicification of the hanging wall. A later deposition of calcite invaded the quartz, and in places it appears that the quartz has been replaced by the calcite.

No. 4 fault strikes N. 50° E. and dips 75° to the southeast. It is intersected on the southerly trend by No. 3 fault and is possibly the northerly continuation of the San Francisco vein beyond No. 2 fault. No. 4 fault with its accompanying zone of brecciation extends to the south end of the C. Q. vein, and to the north it has followed the course of the Don Carlos vein; thus causing brecciation of the wall-rock and in places trituration of the vein-matter to a powder; this has introduced difficulties in mining operations.

No. 5 fault strikes N. 40° E. and dips 45° to the southeast. The fault, so called, is not in reality a displacement—although some late slipping has possibly taken place—but is a mark of contact between the augite-andesite intrusive and the andesite-tuff.

Economic Geology

Yoquivo is essentially a silver-gold district. Silver has constituted about three-fourths and gold one-fourth of the output. Total production to 1908, when the present owners, the Yoquivo Development Co., entered the field, is not recorded, but from calculations based on the area of the old workings, it is estimated production before 1908 about equals that since. From inception of operations by the above-named company the total ore extracted has amounted to about 75,000 tons, assaying gold 0.35 oz., and silver 36.0 oz; or a total of 2,700,000 oz. silver and 26,000 oz. gold. Products shipped have aggregated 377 tons of high-grade ore, 500 tons of concentrate, and 68 tons of bullion. The shipping ore has averaged 11.0 oz. gold and 1130 oz. silver; the concentrate has averaged 26.0 oz. gold and 1600 oz. silver; and the bullion has averaged about 940 fine in gold and silver.

The andesite area of the Yoquivo district is traversed by manifold veins, but only a few have been exploited commercially. Some of those owned by the operating company and named in order of discovery are: the San Francisco, Esperanza, No. 2, C. Q., and Don Carlos. All of these are of the San Francisco mine and the C. Q. and Don Carlos are in the

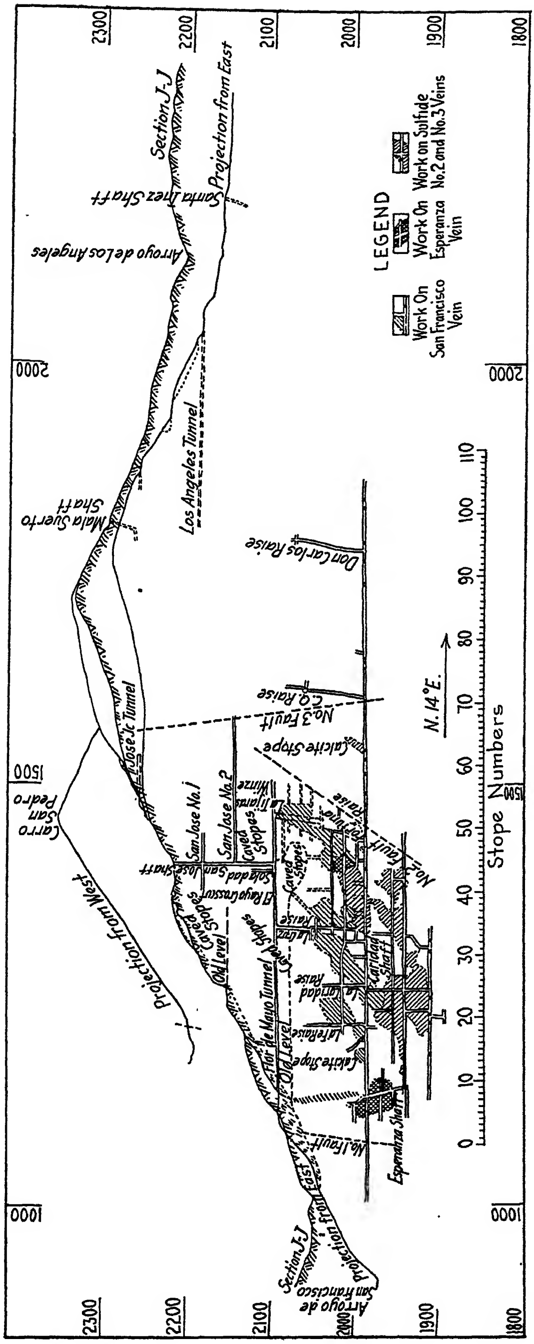

FIG. 2.—SAN FRANCISCO MINE OF YOQUIVO DEVELOPMENT CO., IN YOQUIVO DISTRICT.

northern extension, or new part of the workings. Other veins that have produced some ore are: The Martinica, of the Pertenencia mine, and the Dolores, of the Dolores mine. Considerable work has been done upon veins outside the company holdings, but little ore has been extracted.

The San Francisco, Esperanza, and No. 2 veins are branches of the same ore-channel and have produced the bulk of the ore from the district.

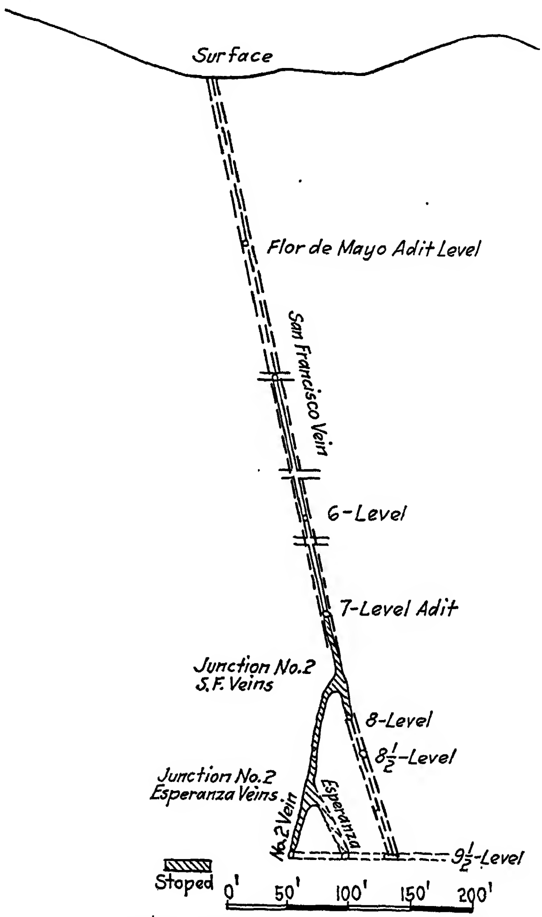

FIG. 3.—CROSS-SECTION OF SAN FRANCISCO MINE.

The C. Q. and Don Carlos are newly discovered veins and have not been fully prospected.

The San Francisco vein strikes N. 14° E. and dips 75° to the east; the width ranges from 4 to 40 ft.; the surface outcrop is strong (see Figs. 2 and 3). The vein is intersected in the foot wall by the Esperanza and No. 2 veins. Both walls in the upper levels are andesite in some of its phases, but lower down the foot wall is porphyry. The filling is com-

posed of calcite, iron-stained (red) gouge, brecciated country-rock, and a quartz seam of varying width; chalcedony is conspicuous. The ore consists of mixed sulfides of copper, lead, zinc, and silver, and their decomposition products. Silver and gold are associated together throughout the quartz. It has been demonstrated that they occur with the copper minerals, other than where native gold and silver are left as disseminations within the gouge. The ore seam is invariably confined to the foot wall. Primary calcite, deposited after ore deposition was complete, is present on the hanging wall as a banded seam, or as cementing veinlets, and never contains ore minerals. The seams range in width from 1 in. to 10 ft., and are characterized by cavities encrusted with protruding crystals of pink and white calcite. In many places large fragments of andesite wall-rock are to be noted floating unsupported in the calcite mass.

The ore in the San Francisco vein is in shoots of differing size. The distance between orebodies is as variable as the shoots themselves. The grade ranges from a few ounces in silver to several hundred ounces; gold is present in an approximate ratio of 1:100 by weight.

Strike and Dip of Esperanza Vein

The Esperanza vein has the same strike as the San Francisco, but the dip ranges from 75° to the west to 75° to the east. The width ranges from a few inches to as much as 15 ft.; widening is usually pronounced at the intersection with the San Francisco vein. The vein-filling is composed of brecciated country-rock and a distinct seam of quartz, which is accompanied by auxiliary stringers, usually parallel, but in places branching from the main course. Gouge, so marked in the San Francisco vein, is lacking. Both walls are highly silicified and extremely hard. The ore, which is associated with quartz, consists of primary and secondary sulfides of copper, lead, zinc, and silver, and at points of oxidation decomposition products from these sulfides (see Fig. 4). Gold accompanies the silver. Except near the southern terminus of the vein and at a few points of contact with No. 2 vein, calcite is absent.

The ore of the Esperanza vein is not continuous, but is less erratic than the shoots of the San Francisco vein and the distances, horizontal or vertical, that are barren of sulfides are relatively short. The grade ranges from a few ounces to several thousand in silver. The gold content is about equal to that of the San Francisco vein. The Esperanza orebody, found on the south end of the vein, contained ore of remarkable richness; a small stope produced about $750,000.

No. 2 vein is little more than a reopening in part of the Esperanza vein. It has approximately the same dip, but a variable strike. The width averages 4 ft. No. 2 intersects the San Francisco 40 ft. below level 7, and comes in contact with the Esperanza vein about 50 ft. lower, where the course of the two coincide for some distance. The vein-filling

is composed of iron-stained brecciated andesite, or porphyry, red gouge, calcite, and usually irregular patches of secondary quartz. The ore includes only secondary decomposition products, deposited as a result of oxidation on the San Francisco and Esperanza veins. The grade is variable. Certain points of tangency with the Esperanza vein were enriched by the precipitative action of the primary sulfides; also at the horizon of the water level high-grade orebodies were found.

All three of the above-described veins are bounded on the south by No. 1, and on the north by No. 2 faults. The distance between is about 1200 ft. Ore has been stoped for about 1000 ft. on the dip.

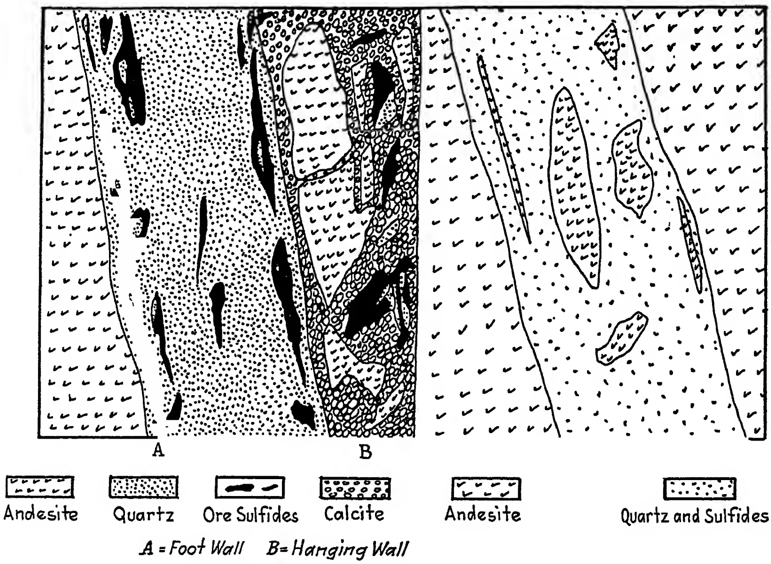

FIG. 4. FIG. 5.

FIG. 4.—ESPERANZA VEIN ON 9½ LEVEL, SHOWING INJECTION OF CALCITE INTO BROKEN HANGING WALL. NOTE WALL FRAGMENTS AND PARTICLES OF VEIN-MATERIAL, INCLUDING SULFIDES FLOATING FREE WITHIN THE MASS OF CALCITE; ALSO TENDENCY OF ORE SULFIDES TO SEGREGATE ALONG WALLS OF VEIN PROPER.

FIG. 5.—A POINT ON C. Q. VEIN SHOWING ANGULAR FRAGMENTS OF ANDESITE WALL ROCK IN VEIN FILLING OF QUARTZ.

The C. Q. and Don Carlos veins are on the adit level north of No. 5 fault. They typify an independent vein-system, in no way coincident with that of the south part of the mine; that is, mineralization is dissimilar, and the origin is hardly to be pronounced cognate. The relative ages of the distinct mineralizations are not apparent. The surface outcrop may be the faulted extension of the main ore-channel from the south; but if so it could only be true in so far as the fissure itself is concerned.

The mineralization of the C. Q. and Don Carlos is analogous, and the general characteristics are identical. The veins strike N. 35° E. and dip 75° to the east. The width ranges from a mere cleft to 10 ft. The wall-rock is augite-andesite from the adit level to about 300 ft. above, where the veins pass through the intrusive contact into the andesite-tuff. The surface outcrop is over 1000 ft. above 7 level. The vein-filling is composed of altered country-rock, gouge, and quartz, which usually occurs as a seam. It is common to find fragments from the wall-rock floating free in the quartz mass (see Fig. 5). The ore is composed of sulfides of lead, zinc, and silver, and native gold, all associated together throughout the quartz.

Late parallel faults have invaded the veins, and in places the matrix has been ground to a powder. The deposit at the present horizon of work is altogether of primary origin.

Mineralization of the C. Q. and Don Carlos veins where encountered on the lower levels is erratic. Samples assay from a few to several thousand ounces in silver; and the gold content is in the proportion of about 1:200 by weight. The vein-system is bounded on the south by No. 3 fault; the northerly limit has not been delineated.

Relation of Ore Deposits to Wall Rock

Thus far in the development of orebodies at Yoquivo there has been no salient relationship established between the character of the enclosing rock and mineralization. Ore has been found in several of the formations and the only obvious reason why it might not occur in any or all, is perhaps that owing to a variation in brittleness, pre-mineral fracturing may not be as complete in some as in others; therefore premising that the tuff should be less favorable for ore deposition, but this has not been confirmed. The veins were formed in fault fissures, and the recognized near-surface type of mineralization precludes any theory of deposition other than that induced by reduction of temperature and pressure; a deposition against which the wall-rock would exert a minimum of influence in precipitation. Without doubt the amount of pre-mineral fracturing affected the shape and extent of the ore deposits, and possibly controlled the arrangement of ore shoots. The irregularity of the C. Q. and Don Carlos veins through the altered contact zone between the andesite intrusive and the tuff illustrates a possibility of restricted deposition due to unfavorable wall rocks.

Genesis of Ore Deposits

The San Francisco vein fracture originally represented a fault of some magnitude, and was the first vein of the system to be formed. The fracture can conceivably be associated with the general uplift of the

region, this by a deep-seated laccolith. Although offering a favorable course for the entry of solutions, no mineralization took place.

The Esperanza fracture was next in order, and was caused by the intrusion of feldspar-porphyry—probably an upper tongue or pipe from the main laccolith. The line of weakness extended up to an intersection with the foot wall of the San Francisco vein, but did not displace it. Later, aqueous hydrothermal solutions ascended by way of the Esperanza fissure, mineralizing its path and, upon continuing upward, entered the San Francisco vein through the foot wall. Thus primary mineralization was effected through the Esperanza vein and along the San Francisco vein, but only from the intersection upward. Both hanging and foot wall are severely silicified and indurated for many feet laterally.

Initial deposition no doubt left concentrated bodies of primary sulfides along the contact of the two veins, and at other points where conditions were favorable; and it only remained for secondary agencies to rearrange the deposits into rich ore shoots.

That primary solutions did not ascend through the larger and more favorable San Francisco vein fissure indicates ore deposition was in some way affiliated with the feldspar-porphyry intrusion in the foot wall; conceivably, the porphyry was not only the cause of the Esperanza fracture, but also came from the same magma from which were evolved the mineralizing solutions.

After primary deposition had ceased stresses along the San Francisco vein were again set up and the vein was reopened. The fault-movement kept strictly to the foot wall and did not disturb the Esperanza vein. This reopening prepared a channel for the circulation of surface waters which resulted in oxidation, solution, migration, and re-precipitation.

Then came the No. 2 fissure, following closely the strike of the Esperanza, and caused by some further disturbance in the foot wall of the San Francisco vein. No. 2 vein received no primary-sulfide mineralization; apparently deposition was effected by downward migration of secondary solutions, which originated in the San Francisco vein, and entered through the intersection along the foot wall.

Last Stage of Ore Formation

Finally, the last stage of ore formation: Due to the reopening of the San Francisco vein, oxidation and solution proceeded, and all parts of the vein were leached; the waters percolated downward along the foot wall and at favorable points precipitation took place. In this way the ore shoots of the San Francisco vein were probably formed. When the migrating solutions reached the intersection with the Esperanza and No. 2 veins, they left the San Francisco vein and followed a new course. Where concentrated bodies of primary sulfides were touched a heavy deposition was effected. This was especially pronounced along the line

of contact and at the water level. The Esperanza orebody in its primary state was probably highly concentrated, and by the aid of secondary enrichment it was transformed into a bonanza.

Although erosion carried away the upper part of the San Francisco vein-system, most of the valuable metal content was saved by solution and downward migration keeping pace with erosion.

This ore deposit is singular in that the San Francisco vein, which has produced the greater part of the ore mined so far, did not tap an ore-magma below, but stole its ore from the smaller and later ore-channel of the Esperanza vein. It is also interesting to note that No. 2 vein, in turn, received its mineralization from the San Francisco vein.

Although possibly constituting the faulted continuation of the San Francisco fissure, the C. Q. and Don Carlos deposit is distinct and independent—in so far as mineralization is concerned—and it is pertinent to assume an unconnected origin. Indeed, the two vein-systems afford little evidence of a cognate relationship.

Apparently the C. Q. and Don Carlos veins were formed by solutions less aqueous in nature than those prevailing in the south part of the mine. The walls are not silicified to any noticeable degree, and the general character of the deposit might signify crystallization from a highly concentrated solution; conceivably of a type that has been called vein-dike injection by Spurr in his "Ore Magmas." The occurrence of the veins in a stock of augite-andesite on 7 level lends credence to the argument for an allied origin with that of the intrusion. That mineralization originated from the porphyry known in the south part of the mine is possible, but it seems germane to postulate a deep-seated laccolith as being responsible for the general uplift of the region, and for having supplied the mineralizing magmas for the entire district; and that ore deposition was effected in conjunction with modified phases of the parent intrusion; manifested in the south by the porphyry, which is an immediate associate of the orebodies found there, and in the north by the augite-andesite, an associate of the vein-system found there.

Secondary Enrichment

Secondary enrichment has played a major rôle in the formation and rearrangement of the orebodies of the south part of the mine. The slipping and faulting, the iron-stained character of the vein, kaolinization, the attendance of chalcedony, malachite and chrysocolla—all secondary products—the occurrence of native silver as casual disseminations through the vein-material, the presence of residual gold in places where silver is absent, all imply secondary action, which is strikingly noticeable at the deepest mine workings, or over 1000 ft. below the highest point on the outcrop.

The orebodies of the San Francisco vein have been enriched and rearranged, and those of No. 2 vein were formed entirely by downward percolating waters. The typical mineralization of the Esperanza vein is fundamentally primary in character, but in places secondary precipitation—mostly as sulfides—aided in forming rich orebodies, such as along the intersection with the San Francisco vein and at the water level.

Within the confines of the mine workings, secondary action has had little or no effect upon the C. Q. and Don Carlos veins. Along the outcrops and in shallow surface excavations, however, the veins do attest leaching, and it is quite possible that at a higher horizon than at present attained, a zone of secondary enrichment might be encountered.

Mineralogy

The Yoquivo ore consists chiefly of metallic sulfides—or their decomposition products—usually in a gangue of quartz, but also as impregnations within the vein material.

The minerals of the San Francisco vein-system, named in order of their preponderance, are: pyrite, copper minerals, headed by chalcopyrite, and including malachite, chrysocolla, covellite, and bornite, galena, sphalerite (usually signifying low gold and silver), argentite, possibly stromeyerite, native silver, and gold. The gold is probably in the native state.

The minerals are much intergrown; this is particuarly true of the Esperanza vein. The rich ore in this system is heavily charged with argentite, or native silver in the oxidized parts; both silver and gold are invariably accompanied by copper. In the San Francisco and No. 2 veins, and in the oxidized parts of the Esperanza vein, the copper minerals are malachite and chrysocolla. The only mineral representing the copper group in the primary mineralization is chalcopyrite. Native silver is wholly secondary.

The minerals of the C. Q.-Don Carlos vein-systems are: pyrite, argentite, stromeyerite, and gold, all intermixed with quartz. Seemingly pure crystals of pyrite assay well in both gold and silver. Late phases of sphalerite and galena have been deposited in parts of the Don Carlos vein. The only copper mineral is stromeyerite. High-grade samples of ore exhibit beautifully crystallized patterns of quartz, heavily charged with argentite and stromeyerite. Masses of almost pure argentite an inch in diameter have been noted. In unique specimens argentite is plated with a thin veneer of pyrite.

Summary and Conclusion

In summarizing the geological features of the Yoquivo district, the following cardinal points seem pertinent:

1. At the close of the volcanic era, during which the region was covered with a thick mantle of Tertiary eruptives, the vein-bearing area was uplifted by a deep-seated intrusion—likely of monzonitic character.

2. Accompanying and succeeding the intrusion, minor faults and fractures were formed; and these lines of weakness provided ore-channels for entry of mineralizing solutions.

3. The mineral solutions probably emanated from upper phases of the primary intrusive magma.

4. During and after deposition of ore, the country was dislocated by sundry faults, which have contributed toward derangement and complication of vein-systems.

5. The San Francisco mine contains two distinct forms of mineral deposition; that in the south, which resulted from an aqueous magma, and is related to the porphyry; and that in the north, which resulted from a dry magma, or possibly a viscous vein-dike injection, and is related to the augite-andesite. Both porphyry and andesite are differentiations of the main intrusion.

Insufficient geologic study has been given the district to fully interpret the phenomena of ore deposition and to work out completely certain structural aspects. This is largely owing to incomplete surface detail and the inaccessibility of the upper part of the mine workings, observation having been confined almost entirely to the level 7 adit and below, in the San Francisco mine.

Geology and Ore Deposits of the Asientos-Tepezala District, Aguascalientes, Mexico

By G. E. Anderson,* Norman, Okla.

(New York Meeting, February, 1926)

The Asientos-Tepezala district is in the north of the State of Aguascalientes, about 30 miles north of the city of Aguascalientes, the capital. The district is reached by a standard-gage railway on the west side through the junction, Rincon de Romos, which is on the main line of the National Railways of Mexico from El Paso to Aguascalientes. The terminal is Tepezala. On the east side a narrow gage connects Asientos with the junction, San Gil, on the main line, Asientos to San Luis Potosi. Tepezala on the west side of the Asientos hills is about 2 miles from Asientos on the east side. The area between the two mining centers is separated by a low subdued divide which forms the central part of the hills.

The district is in a group of isolated hills surrounded by characteristic bolsón plains. The hills rise above the surrounding plains about 6500 to 10,000 ft.; the highest peak is Alta Mira. The two mining districts lie somewhat upon the flanks of the hills at about the same elevation, 7200 ft.

The district is an old one and has been known since early in the 17th Century when the mines were worked by the Franciscan Monks. Their mining operations, which were limited to the oxidized ore above the water level, consisted chiefly in extracting silver from the Santa Francisca vein. To judge from the old workings and the remains of surface plant, the operations were on a large scale. By modern mining operations since about 1900, the various mines of the district have been one of the principal sources of ore for the smelter located at Aguascalientes.

Topography

The topographic features of the Asientos hills are largely controlled by the character of the underlying rocks. During the Tertiary period, the entire group of hills was probably covered by extrusive volcanic material in the form of cones and lava flows. The central part of these have since been carried away by erosion, so that remnants have been left along the borders of the hills. These remnants form rather conspicuous peaks,

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such as Alta Mira north of Asientos, and San Juan Hill in the western part of the hills and immediately east of Tepezala.

The central part of the hills is composed of an extensive area of sericite-schist, which weathers into rounded subdued hills. Its reddish or brownish color easily distinguishes it from the rugged bordering fresh rhyolites or from the limestone found in the district, which weathers to a light gray. The limestones have formed no conspicuous topographic features, but in general are less easily eroded than the schist, and where the limestone rests on the schist, the former tends to form hills. Limestone is found in two rather large areas, one in the Asientos and the other in the Tepezala area, and forms the east and west slopes of the hills. The limestone areas are closely associated with the mineralization of the district and the chief mines on both sides explore veins in it.

General Geology

Schist

The schist, which is the oldest known rock in the district, is of sedimentary origin. It is almost continuously exposed to surface erosion from the Orito mine on the east side to Tepezala on the west, more than 2 miles. It can be traced northwesterly for at least 4 miles. At the Tepezala mines the schist is overlain by later limestones though it continues westerly as far the town of Tepezala, north of the limestone outcrop, where erosion has cut through the overlying limestone. To the north, the schist is covered by limestone and fragments of the lava flows which slope gradually north and end in foothills. On the east side, the schist ends abruptly against a north-south fault which has brought it into contact with the Orito limestone. This fault passes about 250 yards west of the Orito shaft. The contact between the reddish schist on the west of the fault and the light colored limestone on the east is easily traced over the hill west of the Orito mine. To the south, the schist is lost in the plains, where it is covered partly by recent wash and partly by remnants of the rhyolite flows. Within the area of the schist are several well defined fissures filled with white barren quartz, which is also abundantly scattered over its surface.

The schist is predominatingly red through purple, chocolate to brown. It is usually finely laminated and fine grained. Coarser or more arenaceous phases are also present though rather sparingly. Along the planes of schistosity considerable sericite has developed. The planes of schistosity and the bedding planes coincide, for the variegated layers of which the original shale was composed, conform with the planes of schistosity. The layers are usually separated by sericite in the arenaceous parts but in the argillaceous parts the sericite is the chief constituent of the entire rock.

The schistosity strikes north 20° east along its east border but changes gradually to north 45° west along the west side, thus forming a fanshaped strike with the center of the fan to the south. Its dip is strikingly uniform at 70° east.

Alta Mira Limestone

Alta Mira is the oldest limestone in the district. It is a massive, pure, limestone, dense to semicrystalline in character and breaks with a semi-conchoidal fracture. It contains considerable chert in nodules which in the lower part lie parallel to the bedding planes. The limestone weathers light gray to white. It rests unconformably on the schist, which is well shown in the block of limestone immediately east of the Palmira mine. Erosion here has cut through the limestone into the schist below showing the upturned edges of the schist under the limestone, which dips gently to the east. The limestone is more resistant to erosion than the underlying schist so that it forms the low hills of lighter color within the schist area, where isolated blocks are brought down by faulting.

An extensive coating of caliche usually surrounds the limestone blocks within the schist area. This caliche is formed by the solution of the limestone and precipitation of calcium carbonate on the surface, due to the evaporation of the water in the arid climate. In protected areas the caliche has accumulated 20 to 30 ft. thick thereby covering the underlying schist and impregnating it in places to a depth of 10 feet. In a few places the original limestone blocks have been entirely dissolved and eroded away, leaving only the caliche which then forms a white crust over the surface of the schist. This condition has given the erroneous impression that the entire schist area was limestone, which did not look at all improbable when considering the frequently occurring limestone blocks over the schist area, particularly when viewed from a distance.

Thorough examination has shown that the caliche, as well as the limestone blocks from which it has been derived, are in reality superficial. The limestone blocks within the schist area have been brought into their present position by normal faults. The faults have a rather uniform strike of north 50° west and seem to form a part of the series of fault fissures which later were filled with vein materials and are the principal ore bodies of the district. This limestone is not less than 800 ft. thick, which is the same as the block outcropping on the southwest side of Alta Mira. It rests on the schist and is covered by the more recent rhyolite flows which have protected it from erosion.

Orito Limestone

The name is applied to the limestone which forms the country rock for the Orito ore bodies. It is locally known as the "shaly lime." It is composed of alternating layers of limestone and shale, the former rather

uniformly from 3 to 4 in. thick; the latter are 2 to 3 in. thick. The limestone is tilted steeply to the east and sharply folded into small irregular folds which are the most characteristic feature of the entire rock. The folds are usually from 5 to 6 ft. from crest to crest with the shaly members more intensely folded within the larger folds, a feature which is small enough to be seen in hand specimens. In the folding, which appears to be uniform in intensity throughout, the shale strata have been incompetent to withstand the pressure to which the rock has been subjected and have yielded so as to form minute folds within the stratum, while the more resistant limestone strata have been partly broken or bent. This condition of the limestone definitely identifies it wherever it outcrops but also makes it very hard to measure dip and strike. Evidently from many observations, however, the dominant strike is north 20° west and the dip from 60° to 70° east. In the most easterly exposures the dip steepens to nearly vertical as in the arroyos immediately west of the church of Tepezon at Asientos. Along the west boundary the dip is somewhat under 60°. The strike is uniformly north 20° west.

The limestone covers a rather extensive area on the east side of the schist, from which it is separated by faulting. It extends to the north upon the east flanks of Alta Mira and crosses the divide northerly and down into the plains where it is covered by recent wash. It extends under the prominent knob of rhyolite on the north of the Orito shaft, as is shown in the underground workings of the Porvenir tunnel which passes entirely under the rhyolite knob in limestone. On the east side the limestone is found within 100 yd. of the church of Tepezon where it is covered by surface wash and the later rhyolite flows. The south boundary of the limestone is approximately outlined by the San Geronimo arroyo, although a shaly limestone has been reported in the lower workings of the Santa Francisca mine. The thickness of the limestone measured on the surface is at least 1200 ft. but is unknown in the east where it is covered.

Another outcrop of the shaly limestone is in the Tepezala area. This block covers the area from the San Bartolo-Santo Tomas vein system, west and south through the village of El Puerto and westerly over the San Simon group. On the north the limestone has been carried away by erosion, exposing the underlying schist through the village of Tepezala. The limestone in this area dips gently to the east or is often level or but slightly undulating. In general character this block of limestone is similar to the Orito block but has not suffered the folding and crumpling so characteristic of the latter.

Fossils have not been found in any of these sedimentaries so that their geologic age is uncertain. The schist, owing to its position and the reported occurrence of a considerable thickness of Jurassic sediments farther north in Mexico, is provisionally placed in that age. It is

known that a great thickness of limestone was deposited in Mexico to the north of this area during the Comanchian period and the Alta Mira limestone may therefore represent this. Following the deposition of the Alta Mira limestone and preceding the deposition of the Orito limestone, erosion sufficient to carry away the Alta Mira limestone in the Tepezala area must have taken place, as the Orito limestone there rests directly on the schistose shale. This process must have required a considerable time as the Alta Mira limestone is known to be at least 800 ft. thick. The Orito limestone then is referred to the Cretaceous. This places all the sedimentary rocks of the district in the Mesozoic.

Igneous Rocks

Schistose Porphyry

The schistose porphyry is a rhyolitic intrusion into the Orito limestone in the Tepezala area. Its outcrop is about 1,500 ft. wide and forms an elongated mass about 2 miles long from north to south. To the north it ends bluntly against the limestone and to the south it is covered by surface wash. It is the oldest known igneous rock in the district. It has been rendered feebly schistose with its planes of schistosity conforming with those of the sedimentary schist which covers the central part of the Asientos hills.

Santa Francisca Rhyolite

The Santa Francisca rhyolite is an intrusive body of rhyolite, finely prophyritic with phenocrysts of quartz and feldspar. It is in contact with the Orito limestone to the north, in the vicinity of the San Geronimo arroyo west of the church of Tepezon, and is the surface rock north of the Santa Francisca workings for about 300 yd.; its east border is covered by wash and to the south it is covered by more recent rhyolite flows, which form the high ridge south of the Santa Francisca workings. Its entire exposure is due to the erosion which has taken place up the Santa Francisca arroyo and which has removed the later surface flows. An old erosion surface that was developed across it previous to the eruption of the later rhyolites, can be readily traced along the west contact with the overlying flows for about 500 ft.

This intrusive has been found in no other part of the district. Faulting within it has produced a shear zone of brecciated rhyolite along which the Santa Francisca vein and other ore channels were formed.

Andesite Dikes

Within the east area of the Orito limestone are two groups of dikes. The older of these is a dark-colored, fine-grained, rather basic andesite.

It weathers to a light green; on fresh exposure it is much darker and compact, though rather loosely coherent. It outcrops in several exposures in the arroyo between the Porvenir tunnel and the Orito shaft where it intrudes into the schist and along its east contact with the Orito limestone. It is also encountered at the north end of the Orito vein zone, where it was traced from the surface continuously to the 74-m. level, and along this level for about 200 ft. On the 74-m. level it strikes north 25° east and dips 70° south. Fault fragments of a dike of the same character are also encountered on the 94 and the 64-m. levels some distance south of the main body and within the Orito vein zone.

Monzonite Porphyry

This is a coarsely crystalline igneous rock and occurs as a group of dikes in the Tepezala area. The most easterly of these is a narrow dike which cuts the Socorro mineralized zone and can be traced for approximately 600 yd. northerly from the southeast end of the San Juan Hill. It is here entirely within the schist. Another of the same character forms an irregular outcrop in the limestone immediately west of the San Pedro vein and the same body was encountered in the San Pedro workings underground.

Several others outcrop in the San Simon group and other exposures found in the railroad cuts to the west. It is therefore a common outcropping dike rock in the western part of the district. It is later than the mineralization as one of them has cut the San Nicolas vein in the San Simon group and the more extensive of them cuts the Socorro vein system.

Rhyolite Dikes

There are two rhyolite dikes, one of which is prominently exposed about 150 yd. east of the Orito vein and runs south approximately parallel to it for about 100 yd. where it ends. The other outcrop is found at the junction of the Orito and the Minerva arroyos. Both outcrops are similar in character and are closely associated with the later rhyolite flows which cap the Cumbres de Gonzales hill and Alta Mira to the north. They are probably of the same age.

Rhyolite Flows

The rhyolite flows which cap the hills of Alta Mira, form San Juan hill, the ridges south of the Santa Francisca mine, the area of Asientos and a large area several miles east of the town. The lava has probably reached the surface through several conduits of which San Juan hill now remains as an isolated volcanic neck. It is believed that the entire area of the hills was covered by the flows of which the central part has since

been removed by erosion, thereby exposing the underlying schist and limestone. On the hill south of the Santa Francisca mine the flow structure of the lava is everywhere present and west of the town of Asientos in the arroyo, gigantic ropy structures of the lava are exposed. The rhyolite flows contain numerous inclusions of extrusives of both an earlier and the same period of igneous activity, also inclusions of the various sedimentary rocks of the district. These flows represent the last igneous activity in the district.

Volcanic Ash

There are also found at several places on the east side of the district remnants of a cover of very fine volcanic ash. This is at least 15 ft. thick west of the town of Asientos. From its fine character it has probably been carried into the district by the wind from some distant explosive volcano as no evidence of such has been found within the Asientos hills. The volcanic ash forms a soft light-colored rock, which is easily dressed and which was formerly used extensively for churches and the more important buildings in the village of Asientos.

Orebodies of the District

On the east side of the district are the Santa Francisca, the Orito and the minor orebodies of the Minerva and the Palmira-San Geronimo mines. The Santa Francisca is the most easterly mine in the district (Fig. 1). It is a brecciated fissure zone in an intrusive rhyolite, brought about by faulting and consequent shearing. The fissure zone has a width of 40 to 50 ft. It has been traced underground for a distance of about 1000 yd., although on the surface it is exposed for approximately only one third of this distance. It strikes north 50° west and dips 70° west. The brecciated nature of the vein zone has been recently exposed. One of the earlier shafts that penetrated the main orebody has caved to the surface of across the vein zone, thereby giving a cross-section of the latter with fresh exposure which can be studied in the daylight. The brecciated mass within the vein zone is composed of rhyolite fragments similar in character to the wall rock. These fragments in the vein zone have been only partly replaced and the interstitial spaces have been filled by layers of quartz and calcite into which there are interspersed silver sulphides, mostly argentite. Fragments also have been coated with the quartz and calcite. Some of the open spaces between have not been entirely filled, forming small vugs. The banding conforms to the shape of the openings or the fragments of rhyolite which gives to the vein zone as a whole a very irregularly banded appearance and not at all parallel to the walls of the fissure as is commonly the case. The richer part of the vein zone forms a large lense-shaped body in the middle of the vein zone with smaller local bodies north and south, which gradually grade into barren vein matter.

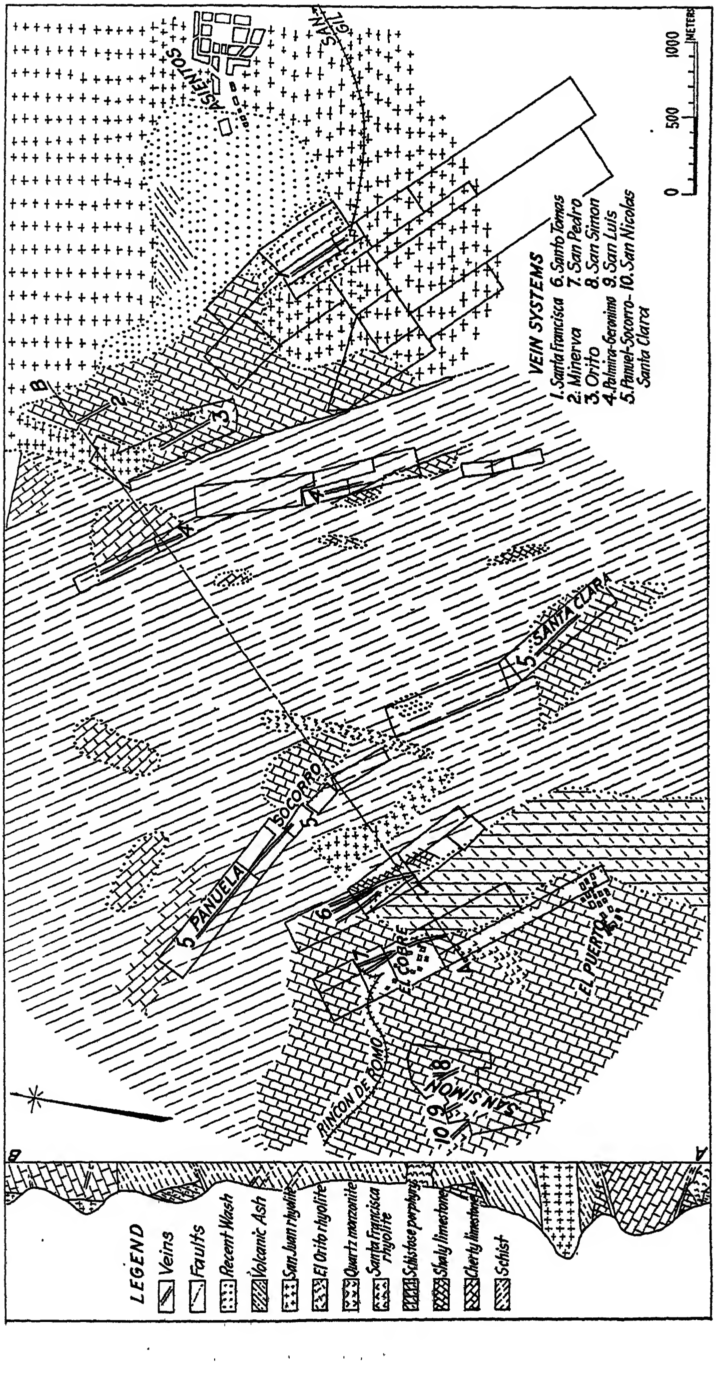

FIG. 1.—MAP SHOWING THE AREAL GEOLOGY OF THE ASIENTOS-TEPEZALA DISTRICT, MEX.

This condition evidently discouraged further exploration so that neither end of the mineralized zone has been reached. The oxidized ore above the ground water level was extracted by the early Spaniards who were unable to cope with the water, however, and hence were forced to abandon the primary ore bodies below. These ore bodies have since been worked until forced to shut down by the political disturbance in Mexico and the lower levels are now flooded and inaccessible.

The Orito Ore Bodies

The Orito ore bodies are found within a fissure or shear zone brought about by normal faulting in the Orito limestone, which forms both the foot and hanging walls. The fissure zone is about 35 yd. wide, being the

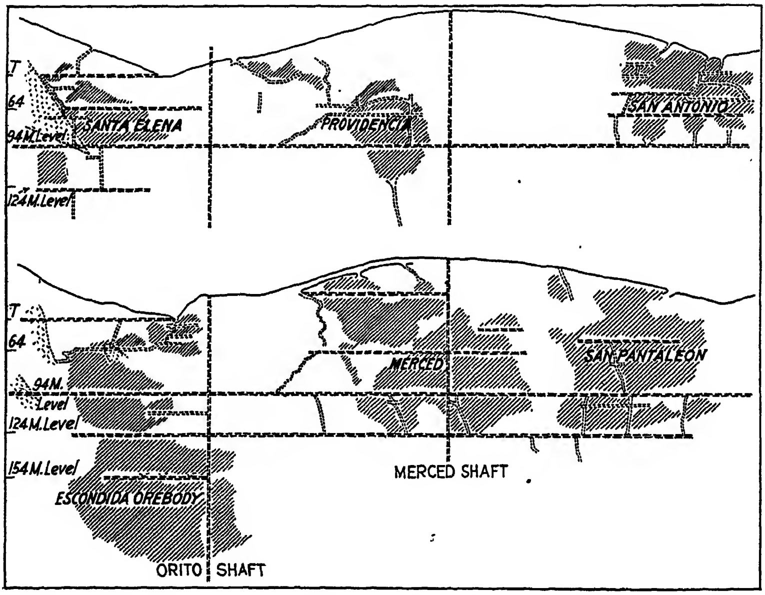

Fig. 2.—Stope map of the Orito vein showing the ore bodies, the principal workings and the andesite at the northwest end (the left, facing northeast).

widest mineralized zone in the district. It is 500 yd. long and terminates at both ends by post mineral faulting. It strikes north 50° west and dips 70° west at the surface. It is about ½ mile northwest from the Santa Francisca vein and but a short distance west of its projected strike (Fig. 1, vein system 3). The surface outcrop is well exposed and forms a prominent exposure of weathered quartz which has been thoroughly leached. In the middle part east of the Merced shaft, the outcrop is covered with recent deposition of caliche. The fissure zone cuts the

limestone both in dip and strike as the latter dips easterly 60° and strikes north 20° west. The wall rock of the fissure are well defined and are composed of unaltered limestone. The entire width within the walls is composed of brecciated limestone of the same character as the walls. It is believed that the brecciated condition of the limestone within the fissure zone was brought about by the faulting which produced the fissure zone itself. The porous condition of the brecciated limestone within the walls formed solution channels through which upward rising solutions precipitated their mineral content and partly replaced the limestone fragments. This replacement was more intense where the currents were stronger, as along the hanging wall, and less so where the solutions permeated more slowly, as in the middle part of the zone and where the limestone has been less replaced and more clearly shows its original fragmentary character. Along the hanging-wall side of the vein zone are thus found three rather distinct orebodies which from north to south are: The Escondida, the Gallineros and the San Pantaleon; opposite these, in the footwall are the Santa Elena, the Merced and the San Antonio orebodies. The Providencia has been brought into its present position by horizontal thrust faulting into the footwall opposite the Merced orebody (Fig. 2).

All of these are primary deposits and probably represent the more porous parts of the brecciated zone where the upward rising solutions gained more ready access. The contact between the orebodies and the wall rock is remarkably sharp, which leads to the conclusion that the rising mineral-bearing solutions were definitely confined within the brecciated zone.

The limestone fragments, which constitute the original filling of the vein zone, have been replaced by quartz, calcite and fluorite with interspersed chalcopyrite, argentite, pyrite and in places sphalerite and galena. The quartz increases with depth, the calcite is more abundant toward the surface, but the fluorite, which is intimately mixed with both the quartz and the calcite, remains approximately constant. The chalcopyrite and the argentite are the principal ore minerals and are closely associated. A small amount of gold showed in the smelter returns.

In the orebodies the outline of the limestone fragments has been somewhat obscured. This is particularly true along the walls of the vein zone. As one follows the development of replacement toward the center of the vein zone one finds more and more limestone fragments which have undergone little or no replacement. The interstices between the fragments have been filled with precipitated quartz and calcite, showing characteristic banding. The centers of the cavities are commonly not filled. The banding conforms to the original outline of the cavities. Similar banding also occurs around the limestone fragments and approximately corresponds to their original shape. Some of the

layers thus formed are very thin and delicate, others are ½ in. thick. The fluorite is intermingled with both the quartz and calcite and does not form distinct layers. The sulphides forming the ore bodies are interspersed through all the layers. The solutions have apparently carried a rather uniform quantity of sulphides and fluorite while there were many changes in the content of silica and calcium carbonate, or in the conditions which caused the precipitation of one or the other of these at successive short intervals, which are recorded in the individual layers.

The Merced shaft was located in the hanging wall about 20 yd. from where the vein zone penetrates the Merced orebody at the 65-m. level and passes into the footwall of the vein zone immediately below the 94-m. level. On the 124-m. level the shaft is about 10 m. from the footwall of the vein zone in the limestone wall rock. This is due to the dip of the vein in the upper part.

Minerva Vein

The Minerva vein is a narrow fissure, 2 to 3 ft. wide, in the Orito limestone. It strikes northwest and dips 40° south. The mineralization consists of coarsely crystalline galena and subordinate amounts of sphalerite. It has been developed to a depth of about 200 ft. where its width is considerably less than in the upper levels. The principal mineral is coarsely crystalline galena. A branch of the vein in the hanging wall, which is approximately vertical, contained oxidized ores which were extracted in the early period of mining in the district.

Palmira and the San Geronimo Veins

The Palmira and San Geronimo veins are narrow fissures in the Jurassic schist. Both of these are located along the strike of the same line of fissuring, the Palmira to the north and the San Geronimo at the south end. Each is located on a fault along which a small block of limestone has been dropped into the schist so the limestone is the hanging wall. Below the extension of the limestone little or no mineralization appears. The strike of the fissure zone is northwest and the dip, steeply east. It is located in the east border of the schist in the central part of the hills.

Tepezala Ore Bodies

The Panuela, Socorro and Santa Clara Veins

The Panuela, Socorro and Santa Clara orebodies occupy a similar fault fissure in the schist, on the west side, as the Palmira and San Geronimo on the east. It is a long continuous fissure which strikes northwest and dips steeply to 70° east. It has been formed by faulting which has dropped limestone blocks into the schist, as at the Palmira just described, and mineralization has taken place where

the limestone blocks form the hanging wall. This fissure zone is traceable for 2 miles and probably extends to considerable depth. It is narrow, however, which is probably due to the less resistant character of the schist that upon faulting has been ground into gouge, thus forming only a narrow zone through which ore solutions reached the limestone fragments where the mineral content was deposited. The deposition here, as in the east side fissure zone in the schist, is limited to the limestone blocks near the surface where they form one of the walls of the fissure. The fissure zone is similar to the Palmira and the origin of the deposits is the same.

San Bartolo, Santo Tomas and Patricinio Veins

West of San Juan hill and along the contact of the schistose porphyry is the San Bartolo vein with its branches, the Santo Tomas and the Patricinio extending north. Where the vein leaves the porphyry to the northwest, it soon disappears in the limestone. The vein can be traced southeast for nearly a mile along the east side of the schistose porphyry. It dips to the east away from the porphyry at an angle approximating 60°. The main ore bodies were found at the northwest end of the schistose porphyry where both walls are limestone. To the south along the porphyry contact, the vein is narrow and only minor deposits have been found.

Over large areas the vein material is coarsely crystalline dark green pyroxene, almost to the exclusion of everything else. Quartz and calcite with small quantities of epidote and green garnet are sparingly present. The principal ore mineral is chalcopyrite with which are associated pyrite and small quantities of silver. The pyroxene has formed in sheave-like crystals normal to the walls of the small fissures, and it commonly forms a coating on opposite walls where the deposition has been disturbed or broken, which leaves a cavity or is later filled with deposits of quartz or, less commonly, calcite. Sometimes quartz and calcite were deposited both before and after the pyroxene. To the northwest where both walls are limestone, calcite becomes more prominent and pyroxene correspondingly less abundant, though some is always found. In the dumps much material is found which shows pyroxene filling, veinlets or sealing of the fragments of a breccia together. Brecciation of the vein material, although not so clearly seen as in the Orito or in the Santa Francisca, has nevertheless taken place extensively. Banding within the pyroxene as well as with the alternating quartz and calcite is rather distinct.

San Pedro—San Maximo Veins

The San Pedro is probably the most extensive vein in the district. It is on the west side of the schistose porphyry and can be traced along

its contact for approximately one mile to the south. It dips southwest so that at its north end the hanging wall is limestone. The vein was originally a fault fissure, shown by shearing and brecciation of limestone and fragments of the porphyry within the vein zone. Many such fragments were present in the dump material. Many of the fragments have been cemented by the gangue minerals, notably by pyroxene. With depth the vein enters the porphyry, is rather narrow and is without commercial ore bodies. Toward the north where it enters the limestone, it branches into two distinct veins, the San Pedro and the San Maximo. At the junction of the two branches and within the limestone the commercial ore bodies are found. The mineralization is very similar to the San Bartolo.

From the mineral content evidently deposition of the vein filling of these two series of veins has taken place at higher temperature than that for the same process on the east side at Asientos. Deposition has taken place in the brecciated fissures or shear zones near the surface. This may be due either to the higher temperature of the mineral solutions with deposition near the surface, or because deposition took place under considerable cover. Owing to the brecciated nature of the original fissure filling, and there is no evidence in the field that erosion has been greater here than on the east side, more probably the mineralizing solutions at the time of the deposition were of higher temperature and deposition took place comparatively near the surface. Although the mineral assemblage of both the San Bartolo and San Pedro veins is characteristic of contact deposits, there is no evidence in the field of other contact effects.

San Simon Group

The San Simon group is about ⅓ mile west of the San Pedro vein farther out on the west flanks of the mountains. This group contains three mineralized fissures all in the shaly limestone of the same block which forms the wall rock of the San Pedro vein. From east to west the veins are the San Simon, the San Luis and the San Nicolas. All strike approximately north 45° west.

The San Simon, which has proved the most important in ore production, dips 45° southwest. It divides into two branches at the north end, but the western branch extends only a short distance beyond the junction and is barren. At the junction the vein is nearly vertical and the largest and most productive ore body is located here. This consists of chalcocite probably derived from primary chalcopyrite. Some distance south of the junction the vein filling is coarsely crystalline calcite with finely disseminated argentite which has stimulated exploration in this direction.

In the north end the wall rock on both sides of the vein has been intensely silicified and the alteration has taken place with perfect pres-

ervation of the original laminations of the limestone. The laminations can be traced from their completely silicified phase in the walls of the vein into limestone at 20 to 25 ft. from the vein. The replacement has taken place without change of volume.

The San Luis and the San Nicolas veins have many features in common. Both dip southwest at approximately 70° with wall rock of the limestone. Both are narrow fault fissures. They have similar mineralization; chalcopyrite predominates and with it are associated minor amounts of pyrite and sphalerite in a calcite gangue. The limestone of the wall rock has been selectively replaced along certain laminae for 15 to 20 ft. from the vein.

In addition to these are several minor fissures of which the Fortuna vein within the schistose porphyry and the Veta Negra group along the west contact and to the south of the schistose porphyry may be mentioned. The former is important merely because it indicates that the fissuring has taken place at some subsequent period to the metamorphism which produced schistosity in the porphyry and the adjacent schistose shale.

The Veta Negra group is located off the map (Fig. 1), south along contact of schistose porphyry.

Regional Metamorphism of the District

At an early period in the geologic history of the district, the rocks then present were subjected to a compressive movement which resulted in a northwest-southeast trending strike of all the rocks present. The result of the compressive movement is clearly shown in the sericite schist, which is the earliest known rock in the district, as well as in the folded and foliated condition of the Orito limestone on the east side of the schist and in the rhyolite intrusive, locally known as the schistose porphyry in the Tepezala area.

As has already been noted the sericite schist covers a very extensive area, at least 2 miles wide from east to west and 4 miles long to the northwest-southeast. The strike of schistosity is rather uniformly northwest. East of the schist at Asientos is the foliated and crumpled Orito limestone with the axes of folds trending northwest throughout its entire area. It extends east of the schist nearly ½ mile to near the church of Tepezon, west of Asientos. In the Tepezala area there is a comparatively narrow outcrop of a rhyolite intrusive trending northwest which has been rendered schistose with the strike of schistosity also northwest.

Following the intrusion of the rhyolite porphyry and probably preceding its complete consolidation, the compressive movement took place which rendered all these rocks schistose. It does not seem likely that the metamorphism in the district was local, as has recently been advo-

cated by Spurr,[1] but that it affected all the rocks then present in the district and covered a large area in the center of the Asientos hills. This area consists of the outcropping of the sericite schist, the Orito limestone and the schistose porphyry. (See Fig. 1.)

Origin of the Veins

Following the compressive movement of the district which has just been described, the area was subjected to tensional stresses in the opposite direction from which compression had formerly taken place. The tensional stresses were eased by the formation of series of normal faults and shear zones, with strike normal to tension and therefore also in a northwest-southeast direction. Some of the faults dip northeast, others dip southwest, all about 70°. The fault fissures are readily divided into two groups, namely, simple fault fissures which form narrow fissure veins, when filled with vein matter, and shear zones from 20 ft. to more than 100 ft. wide. To the former group belong the Palmira—San Geronimo group, the Panuela—Socorro—Santa Clara group, the veins of the San Simon group and the Veta Negra group. To the latter, or shear zone series, belong the Santa Francisca, the Orito, both on the east side of the district, and the San Bartolo and the San Pedro vein systems on the west side of the district. In the latter group the shear zones have been brought about by normal faulting within which the country rock has been brecciated throughout the entire width of the shear zone.

The brecciation of the country rock within the shear zone has caused an increase in volume of the brecciated materials which, when limestone or rhyolite, has been competent to keep apart the walls of the shear zone and thus developed porous zones that formed channels for upward rising mineral solutions. These wide veins or shear zones were thus formed by faulting and the walls were kept apart or perhaps partly forced apart due to the increase in volume of the material of the zone on being broken into fragments. This increase in volume of materials on being broken seems a self evident fact. It is taken advantage of in various mining operations as overhand stoping and must always be taken into consideration in the filling of stopes. Large fragments of country rock were in fact observed in the Santa Francisca vein,[2] by Spurr, who[3] also notes that inclusions of wall rock exists in the San Pedro and the San Bartolo veins at Tepezala. It does not seem necessary, therefore, so far as the veins are concerned in this district, to say[4] that the rather indefinite force of "Telluric pressure of the vein-forming solution" kept the walls apart during the deposition of the vein materials.

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[1] J. E. Spurr: "Ore Magmas." McGraw-Hill Publishing Co., New York, 1923, I, 276.

[2] *Op. cit.*, 283. [3] *Op. cit.*, 280. [4] *Op. cit.*, 281.

In a brecciated zone thus produced by faulting, more or less open spaces would occur which would probably be arranged somewhat parallel to the main walls of the zone. In such open spaces banding would result which would appear to be parallel to the walls. Also, in the more intimately brecciated parts, replacement may not be complete, in which event fragments of wall rock will be found which are not entirely replaced although completely surrounded with vein matter. In a brecciated vein zone, such fragments could hardly be oriented with the wall rock of the vein nor could they always be expected to be rounded but may also be angular, as has recently been pointed out by Bateman.[5] It should also be noted that the wide veins of the district are found only in the more competent rocks such as the limestone and not in the softer schist areas where faulting of equal magnitude is present. In the schist the fault fissures are narrow and the fault movement has yielded finely comminuted materials now forming gouge along the vein walls. If the veins are due to vein-dike intrusions, it would seem that the widest veins might be expected to form in the more readily yielding rock, such as the schist.

The mineralization, furthermore, in a vein dike could hardly be influenced by the wall rock to any great extent such as is known to exist in many wide veins and to which the Tepezala veins are no exception. This is strikingly shown in the San Pedro vein in which the ore bodies are found at the north end of the fissure only where the walls are limestone, whereas it is barren to the south where it cuts the schistose porphyry. Similarly selective precipitation has taken place in the fissures that cut the schist where ore bodies are found, only where limestone is present and forms one of the walls. At depth below the extension of the limestone, the ore deposition has not taken place as is shown in the Panuela, the Socorro, the Santa Clara and the Palmira. In continuations of the fissures into the schist where both walls are of this material, the veins are barren.

Chronological Sequence in the District

1. Deposition, probably during Jurassic, of shales of great thickness which form the oldest known rocks in the district.
2. Erosion, followed by submergence and deposition of cherty nonfossiliferous limestone at least 800 ft. thick, resting unconformably on the shales; the Alta Mira limestone; probably Comanchean in age.
3. Deposition of shaly nonfossiliferous limestone about 2000 ft. thick; the Orito limestone; believed to be of Cretaceous age.
4. Intrusion of the schistose porphyry at Tepezala.

[5] A. M. Bateman: Angular Inclusions and Replacement Deposits. *Econ. Geol.* (1924) **19**, 504.

5. Intrusion of the Santa Francisca rhyolite.

6. Regional metamorphism which affected all the rocks in the district then present and which resulted in the development of a northwest-southeast trend in all the rocks of the district. This movement developed schistosity in the shales and the schistose porphyry and sharp folds in the shaly limestone.

7. Following the compressive movement, tensional stresses developed in reverse direction to the earlier compression, resulting in the formation of series of normal fault fissures and shear zones, normal in strike to the tension and hence parallel to the strike of schistosity.

8. A period of mineral-bearing ascending solutions which entered the fissures and brecciated zones where they were precipitated as

(*a*) The chalcopyrite, etc. with pyroxene gangue of the Tepezala area.

(*b*) The chalcopyrite with quartz-calcite-fluorite gangue of the Orito vein.

(*c*) The chalcopyrite replacement of the San Nicolas and San Luis veins.

(*d*) The San Simon silver-bearing veins with calcite gangue and locally copper-bearing with siliceous gangue.

(*e*) The Santa Francisca silver-bearing veins with quartz and calcite gangue.

9. Intrusion of quartz-monzonite dikes in the western part of the district.

10. Extrusions of rhyolite flows which probably covered the entire area.

11. Erosion and part removal of the latter flows which exposed the older underlying rocks and the outcrops of the veins in them.

12. The present formations.

Geology of the Zaruma Gold District of Ecuador*

By Paul Billingsley, Portage, Wash.

(New York Meeting, February, 1926)

In their course across Ecuador, the Andes fail to show the mineral wealth with which they abound in Peru, Bolivia, and Chile. This may well be due merely to the concealment of recent volcanic ash and dense tropical jungle, but at any rate Ecuador has lacked the stimulation to development and progress afforded to the southern countries by the mines of Cerro de Pasco, Potosí, etc. There is, however, in Ecuador, near its southern border, one mining district that, in recent years under North American management, has become a steady producer of mineral wealth and a community in which the sanitary and mechanical elements of modern life are displayed, for the first time, to the Ecuadorians of the lowland forests.

The mining district of Zaruma is situated in the province of El Oro, near the southwestern corner of Ecuador, about 50 km. (31 mi.) from the coast. Pizarro's first landing on the mainland was at Tumbez, at the mouth of the river that drains the Zaruma area; and the gold so abundantly found in this Indian town was soon traced to its source. In 1549, Mercadillo, one of the original band, reached Zaruma, having worked his way up-stream through canyons that are now regarded as impassable.

The Spaniards proved themselves efficient prospectors, finding practically every orebody in the district as yet discovered; and, working with impressed Indian labor, they were able to mine ore of lower grade than can now be profitably extracted, even with modern equipment.

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* This paper is presented by permission of J. W. Mercer, general manager, South American Development Co., and owes much to his revision. The general subject of mining at Zaruma and the history of modern operations there were described by him in a paper "Mining in Ecuador," before the Second Pan-American Scientific Congress, at Washington, D. C., Dec. 27, 1915. The geology of Ecuador has been summarized by W. A. Wolf in *Engineering & Mining Journal*. The general geology of Zaruma has been described by John Baragwanath, in the *Columbia School of Mines Quarterly*, and is discussed in Singewald and Miller's "Mineral Deposits of South America." The mining methods were described by Rudolph Emmel in *Trans.* (1925) **72**, 447.

In his study of the geology of the Zaruma district, the author owes much to the officials of the South American Development Co., who participated in the underground and field work and made many helpful suggestions.

Prof. R. J. Colony, of Columbia University, made a petrographic report on a suite of selected specimens, which contributed greatly to the accuracy of our knowledge.

The outcrops were first attacked by sluicing, water being led around the steep hillsides as far as necessary to reach the uppermost portions. Soil and decomposed vein matter were washed down to feed small stamp mills, with stone shoes and mortars, and the crushed material passed over riffles. These crude methods no doubt effected but a poor recovery, even in the oxidized, free-milling portion of the veins. They were continued in many cases until a trench 50 or 60 ft. deep had been washed into the hillside along the course of the outcrop. In the gulches below these washes, the old stone stamps are still found.

When a vein could no longer be exploited in this manner, underground workings were driven and the ore stoped. Although the steep hillsides afford room for many tunnel levels, the Spaniards apparently did not conceive the idea of utilizing a lower outlet for their ore. Their workings usually go down on an irregular incline until water level is reached; and the stopes and raises form an amazing labyrinth. The fact that ore was carried out in skins and baskets on the backs of Indians, no doubt explains the indifference shown toward making the workings level. At water level, the Spanish operations ceased, as they had no means of pumping water or treating base ore.

As the easily reached free-milling ore became exhausted, therefore, the Spanish exploitation declined; and during the disorders of the war of South American independence, in the early part of the nineteenth century, the mines were practically abandoned. The workings became the nesting place of myriads of bats and, in many cases, the portals were completely hidden by the dense tropical vegetation.

From this condition the district has been rescued by the efforts and energy of foreigners. It has not been an easy struggle. Inaccessibility and tropical diseases have multiplied the ordinary hazards of development. The first company, a British organization, gave up the attempt after much work and expenditure. The real merits of the district, however, could not be ignored and, in 1896, the present owners secured control of the more important mines.

The subsequent years have seen steady progress in development and metallurgy, combined with improvement in sanitation and living conditions—the latter occupying the attention of the management to an extent hard for the northern engineer to appreciate. Just as the success of the Panama Canal depended on medical and diplomatic talent no less than on engineering, so the profitable development of the Portovelo mines has necessitated the solving of many problems not normally inherent in the mining industry. An unskilled and almost childlike type of native laborer has been guided and trained into a fairly good miner. Disagreements have been settled with good temper and absence of violence. Malaria has been almost eradicated by widespread clearing of brush, draining, and the use of screens; dysentery and typhoid by care in the use

of drinking water and by the introduction of sewage systems. The white employees have comfortable homes and all the recreations of a normal American community. The living conditions of the native employees have been vastly improved. At the end of two days' mule ride over the jungle trail, which is the sole means of communication with the coast, the sight of the pleasant modern camp of Portovelo, with its typical American comforts, fills the visitor with amazement at the achievement represented.

General Geology

The superficial conditions in Ecuador are so strange and exotic to a North American geologist that he is surprised to find the fundamental economical geology of the Zaruma district conforming closely to the type of Rocky Mountain mining camps. The differences are obvious: the covering of tropical vegetation, the more deeply weathered soil, the paucity of fresh outcrops; but the resemblances, when the geology is once worked out, are no less striking. Thus, the ore at Zaruma occurs in fissure veins, which form a linked-vein system in and adjacent to a granitic intrusive cupola. A certain portion of the intruded country rock is most favorable to the formation of these veins; the ore is largely confined to this portion. The minerals in the veins are deposited in a series of primary zones, the hotter zones closest to the intrusive. Post-mineral faulting is common and the faults tend to follow the lines of weakness represented by the earlier vein-filled fissures. Secondary enrichment is present.

All of these statements could be made with equal correctness about many of the great mining districts of the Cordilleran region—the Coeur d'Alenes, Butte, Park City, Bingham, Tintic, and Bisbee, for example—and one is tempted to diverge into a study of the similarity of ore deposits, which might be fully as profitable as the more usual studies of the classification of ore deposits.

The Zaruma district lies in southwestern Ecuador. Here the Andes, which farther north consist of two definite parallel ranges, each crowned with volcanoes so high that even under the equator they stream with glaciers, break up into an irregular knot of mountains of only 11,000 or 12,000 ft. elevation. The western foothills of this knot extend well toward the Pacific, narrowing the coastal plain to a few miles in width. The rivers have trenched deep valleys into this foothill zone and one, the Rio Tumbez, with its tributaries, has excavated a great amphitheater well back toward the main range. Zaruma lies in this amphitheater, on a ridge between two of the main streams—Calera and Amarillo—that go to make up the Tumbez. Portovelo, the mining camp, is in the Amarillo valley at the foot of this ridge. The river bottoms are at an elevation of 2000

to 2500 ft.; the ridges rise quickly to 4000 or 5000 ft., and a few miles eastward climb to a height of 10,000 ft. or more.

Country Rocks

The rocks that compose this area of rugged country are, with few exceptions, monotonous in their uniformity. Greenish volcanic flows, greenish volcanic breccias and agglomerates, and dark sills are the common types throughout the Zaruma drainage area. In composition, they are andesites and augite-andesites, and the deep tropical weathering reduces them, at the surface, to brown and yellow clayey soils. Originally these extrusive volcanic rocks must have lain nearly horizontally, upon a floor of earlier sediments and schists (areas of which can be seen here and there in upfaulted blocks), but they are now tilted to the southwest, with dips of 35° to 45°. These dips are steeper than the general slope of the surface in the same direction, so that erosion has stripped the uppermost beds from the eastern ridges and exposed successive belts of the underlying series, representing an enormous thickness of volcanic debris.

In the Zaruma district itself, the fragments in the agglomerates are of moderate size, seldom exceeding an inch in diameter; but a few miles west they become, in some of the beds, of great size (several feet across), and the proximity to the vents thus indicated is confirmed by the discovery of several volcanic necks in this area.

As tilted, the agglomerate beds have a northwesterly strike, and the belts of slightly different colored soil corresponding to the successive layers can be traced across country in that direction. They show, ordinarily, little or no evidence of mineralization in the economic sense.

Intercalated among these layers of greenish breccias and flows, however, is a different formation, which is at once noticed because of the warm reddish soil into which it weathers. The belt of this red soil extends through Portovelo itself and northwesterly for 20 km. (12½ mi.), where it is lost in the tropical forest on the crest of the Cordillera de Dumari. This formation is composed of medium-textured andesite sills, of greenish-gray color and, usually, prominent plagioclase and hornblende phenocrysts;[1] we have named it the Portovelo.

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[1] Thirty-four type specimens from the Zaruma district were sent for petrographic study to Prof. R. J. Colony, of Columbia University; this and the following petrographic descriptions are from his report.

Normal Portovelo andesite—a felsite rock originally porphyritic with xenolithic inclusions. The essential minerals are moderately basic plagioclase and pyroxene; the accessory hornblende, magnetite, apatite, and orthoclase. The rock (in the specimens from the mines) has been hydrothermally altered, the solutions selecting the ferromagnesian minerals, which are replaced by chlorite, carbonate, epidote, talc, and leucoxene. Along slight fractures epidote, pyrite, and carbonate have been introduced. The rock is classed as an andesite porphyry.

The Portovelo series divides the green volcanic rocks into two parts: That which underlies it, and outcrops to the east, we have called the Muluncay series; that above and to the west, the Faique series. Fig. 1 shows the distribution of these groups in the Zaruma district, and the cross-sections, Fig. 2, make clear their relationship to one another. The Portovelo andesite is the favorable stratum of the country rocks in which the orebodies have formed.

Intrusive Rocks

The Portovelo formation is not intrinsically ore bearing. It does not contain important veins throughout its course across the region; only in the Zaruma district proper. Its productivity in this area is closely associated with, and dependent on, the group of intrusive rocks there found.

These intrusive rocks occur in a north-south zone, which extends from Portovelo to and beyond Zaruma; see Fig. 1. This course takes them across the outcrop of the Portovelo formation, so that at the southern end they are in the upper Portovelo and Faique rocks; while at the northern, they are in lower Portovelo and Muluncay.

Three distinct types of intrusive rock are found; first, coarse granitoid rocks of granodiorite composition;[2] second, medium-textured granular quartz-monzonites,[3] heavily pyritized; and third, medium to coarse-textured basic rocks (labradorite andesites),[4] almost basalts. Their sequence in age is in the order named, the basic rocks being the latest and cutting all the others, but all are very close together in point of time.

The first type I have called the Castillo intrusive; two other stocks, the Tres Reyes and Diez Vetas diorites, are allied. The second type is the Soroche intrusive, to which the Sesmo intrusive is closely akin. The third is named Agua Dulce and Curipamba in its two principal areas.

The distribution of these intrusives can be seen in Fig. 1. They occur as more or less isolated stocks, apparently cupolas from a deeper seated mass below. Such a mass is indicated by the fact that to the north and east of the district large areas of similar intrusives occur. The

[2] Professor Colony says: Essential minerals are acid plagioclase, quartz, soda orthoclase, and ferromagnesians, probably originally biotite(?). The quartz occurs in the ground-mass aggregate and also as corroded phenocrysts. Sericite, carbonate, chlorite, and leucoxene occur as secondary alteration products. The rock is classed as a quartz-keratophyre.

[3] Professor Colony finds that the specimens from this formation are predominately quartz-monzonite, although some range from quartz-porphyrys and granophyres to monzonites without quartz. The essential minerals are usually orthoclase, acid plagioclase, quartz, and altered ferromagnesians; apatite is a usual accessory, and pyrite has always been introduced.

[4] Professor Colony says: These rocks contain as essential minerals labradorite and light colored pyroxene, with accessory olivine and magnetite. Serpentine, carbonate, leucoxene, and quartz are the usual alteration products, and some quartz, pyrite, and carbonate have been introduced by mineralization.

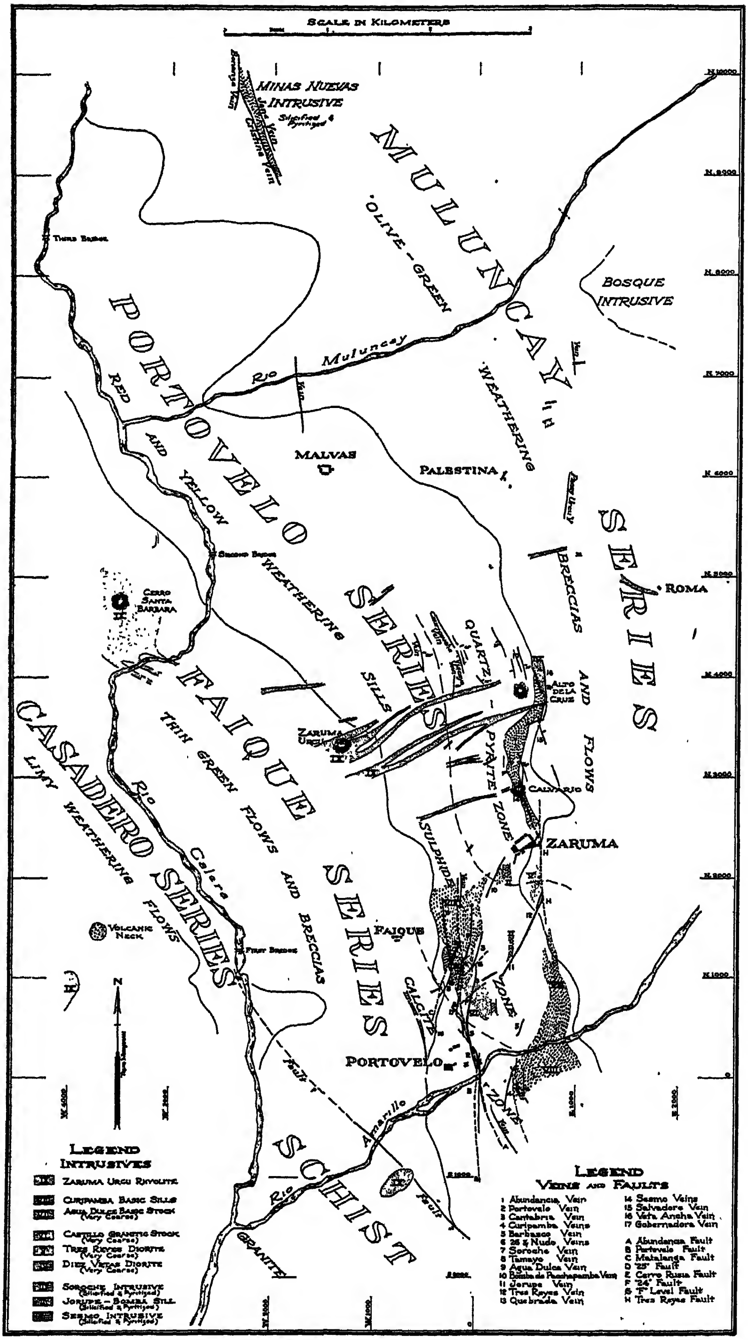


FIG. 1.—GENERAL GEOLOGICAL MAP OF ZARUMA DISTRICT.

Bosque intrusive, shown in Fig. 1, is the western edge of this area, which is probably continuous across to the Amarillo River. At Bosque, it is a quartz-diorite porphyry;[5] on the Amarillo a granodiorite,[6] with segregations of alaskite. The various slightly different types found in the Zaruma district may well be varying segregations from such a common parent batholith.

A similar granitic batholith is found about half-way to the coast, near La Chonta. It has, likewise, acidic phases and is traversed by veins of vitreous quartz and pyrite.

A fourth type of intrusive has been left for separate discussion, as it is separate in occurrence and type. This consists of acidic rhyolite dikes and stocks, which apparently formerly fed a widespread overlying flow. Remnants of the latter occur as capping on some of the higher hills, while nests of great residual boulders in certain ravines testify to its former greater extent. The most important stocks in this district are at Zaruma Urcu, about 2 km. (1.2 mi.) northwest of Zaruma, and at Cerro Santa Barbara, an equal distance beyond. Others are found at points entirely out of the Zaruma area. These stocks are the foci of innumerable dikes. From Cerro Santa Barbara and Zaruma Urcu, many radiate out across the northern part of the Zaruma district, keeping a general easterly course and getting finer and more acidic with distance from their source. Some of the smaller branches occur well down toward Portovelo, where end-product vein dikes of fine quartz occupy some of the fissures. So far as can be seen, these rhyolite intrusions have no influence on the mineralization of the district. They are only found in proximity to orebodies in the Portovelo-Zaruma area, the rhyolite elsewhere showing no evidence of having stimulated valuable mineralization.[7] They may be considered as a generalized acid end product of the underlying batholiths, while the vein filling is a specialized end product derived from local cupolas.

Fissuring

All the intrusives no doubt induced a certain amount of fissuring, but the maximum stresses and resultant formation of fissures occurred during

[5] Professor Colony finds it a moderately coarse porphyritic rock with andesine, quartz, and altered pyroxene for essential minerals. Accessory are orthoclase, olivine, and ilmenite. Secondary alteration products are actinolite, epidote, carbonate, serpentine, and leucoxene, while pyrrhotite (?) has been introduced by mineralization.

[6] The specimen from the Amarillo is described, by Professor Colony, as coarsely granitoid in texture, with essential acid andesine, quartz, orthoclase, hornblende, and biotite, and accessory apatite and magnetite. Plagioclase dominates over orthoclase and quartz is abundant. Secondary alteration is slight, but has produced some chlorite, epidote, sericite, and leucoxene.

[7] Professor Colony finds the rhyolite flow to be a devitrified rhyolitic vitrophyre; that is, a volcanic glass subsequently altered. It has quartz phenocrysts and feldspars, both plagioclase and alkali, and a little altered biotite. It is slightly spherulitic. He agrees that the various quartz dikes could well be end products from such a magma.

the intrusion of the big granitic batholith and its cupolas. As these cooled, the entire region (country rock and intrusive stocks together) was forced to adjust itself to a new basis of stability; an adjustment that

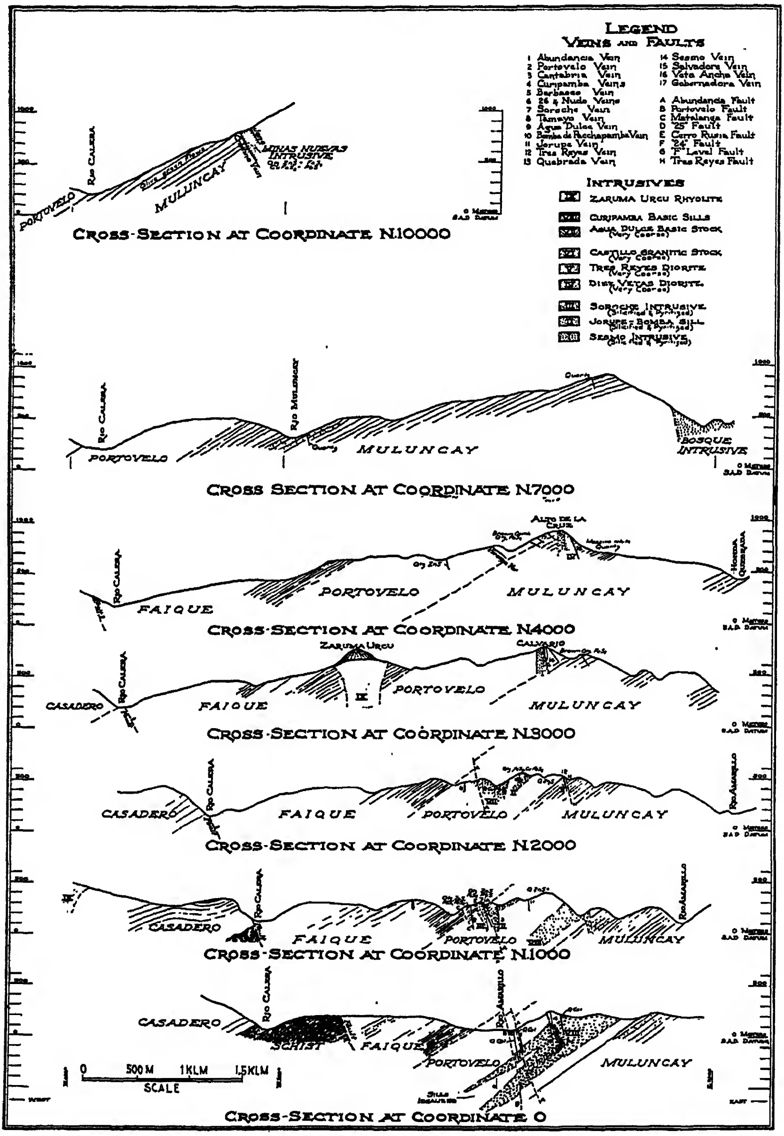


FIG. 2.—GEOLOGIC CROSS-SECTIONS OF ZARUMA DISTRICT, LOOKING NORTHWARD.

necessitated the tilting, warping, and shrinking of all the formations. The different members of the country rock reacted differently to these forces.

The Faique and Muluncay formations are soft bedded breccias and flows with thin flow lines; under stress they yielded and bent, as a shale acts, forming few fissures.

The Portovelo formation is much more massive and rigid. Like glass or quartzite, it could not bend much and so was shattered, the cracks forming a fissure system across the formation from south of Portovelo to well north of Zaruma; see Fig. 3. The shape of this system is typical of the fissuring that results from regional stresses developed by igneous intrusions. The early vein systems at Butte, Mont., and the veins at Park City, Utah, conform to the same type. Other instances will occur to all familiar with Rocky Mountain mining camps.

Mineralization

These fissures, formed in the intruded Portovelo andesite and also in the upper cooled portions of the intrusive stocks, became in turn the natural channels for the circulation of mineralizing solutions. Almost every intrusive igneous rock, as it cools and crystallizes, forces out from its mass hot siliceous alkaline solutions, which carry sulfides of the common metals. The intrusive cupolas in question conformed to this rule and, from some deep-seated source, mineralizing solutions were injected into the fissure system. From the resulting vein fillings, it can be seen that these solutions carried silica, iron, copper, zinc, lead, lime, and manganese as well as gold and silver, which were deposited at different points in the vein system.

The mineralization was long continued. The initial surge of solutions was most intense and widespread, causing great alteration in the wall rocks beside filling the veins with quartz and sulfide minerals. The Soroche quartz monzonites, in the Portovelo area, and the allied Sesmo intrusive near Zaruma, were attacked to the greatest degree by the alteration processes and seem to be the foci of the mineralization. Silicification, pyritization, and the development of sericite are the principal changes effected by these early solutions.

The second period of mineralization was cooler, weaker, and more localized. Many of the smaller veins, closely sealed by the first vein filling, were not reopened and received none of the second generation. The main veins, however, lying in the natural lines of weakness of the country, were refractured from time to time and were available for the circulation of the later solutions. The second generation of vein filling is found in these main veins. It occurs as distinct later bands of white quartz, quartz and calcite, or calcite alone, or as a cement of these minerals including fragments of the earlier filling of quartz and sulfides. This second generation is found mainly at the southern end of the main vein system and is entirely lacking in the northern area.

Still a third generation of vein minerals is sometimes found—always in the main vein near the southern end. It consists of amorphous silica, usually dark, with chlorite and very fine pyrite, and occurs as a cement around fragments of second-generation quartz and calcite. The principal faulting had begun before these last solutions had stopped circulating, and they are occasionally found replacing crushed rock and gouge along the fault lines.

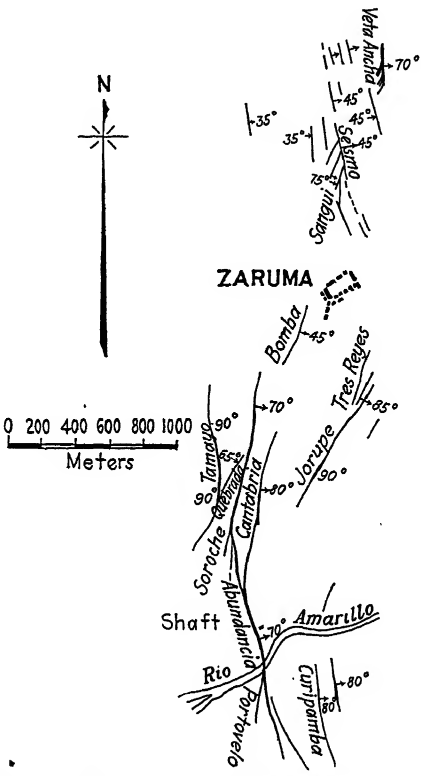


FIG. 3.—VEIN SYSTEM OF ZARUMA DISTRICT.

MINERAL ZONES

The foregoing outline shows that the different minerals in the veins are distributed in a rather definite manner, one set predominating in one section of the district, another set in another section. The veins north of Zaruma, for instance, contain only quartz, light colored pyrite, and a very little chalcopyrite. One, the Veta Ancha, is massive quartz alone. Southward toward Portovelo (see Fig. 1), one finds the proportion of sulfides increasing. At first, as in the Bomba de Pacchapamba workings and the lower levels of Agua Dulce mine, chalcopyrite is the predomi-

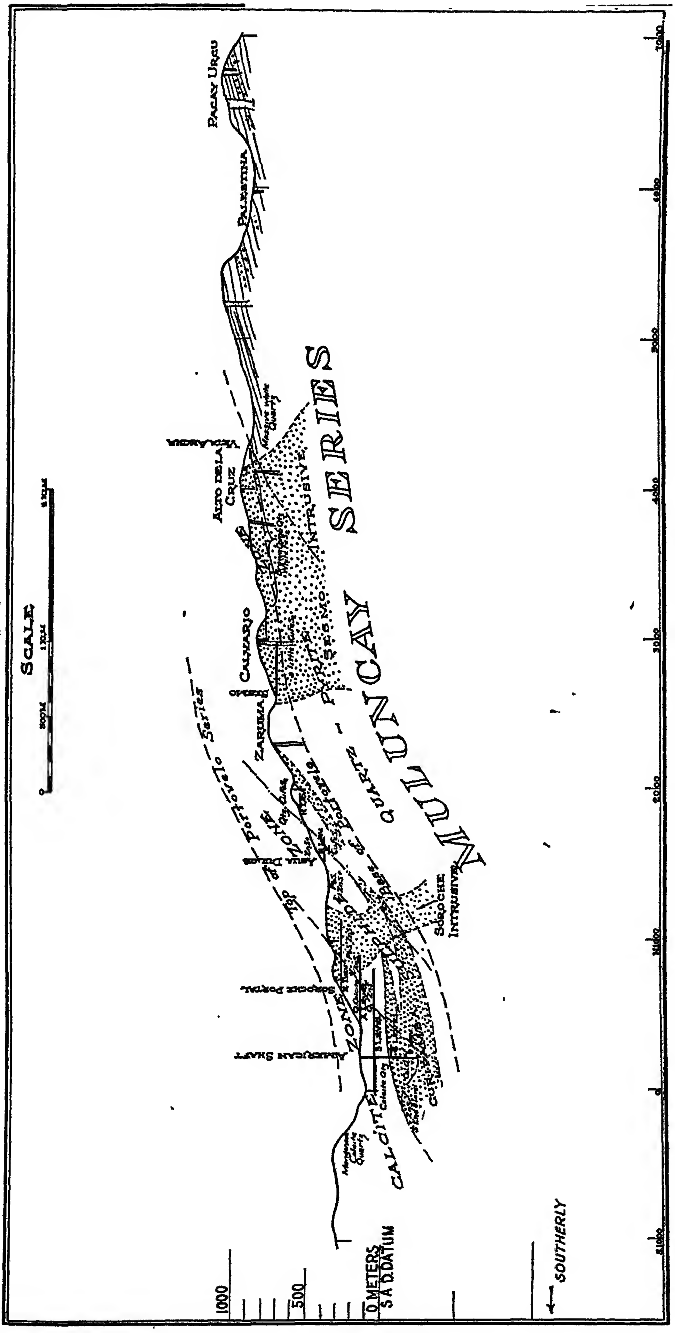


FIG. 4.—NORTH-SOUTH SECTION ALONG GENERAL VEIN SYSTEM OF ZARUMA DISTRICT, LOOKING WESTWARD.

nant sulfide. Farther south, in the Tamayo, Soroche, Cantabria, and Jorupe veins, sphalerite predominates, and all the sulfides are abundant—pyrite, galena, and chalcopyrite being found with the sphalerite in a typical "complex ore" aggregate. The miner acquainted with the zinky ores of Park City, Butte, and the Coeur d'Alenes would feel quite at home with this ore of the sphalerite zone.

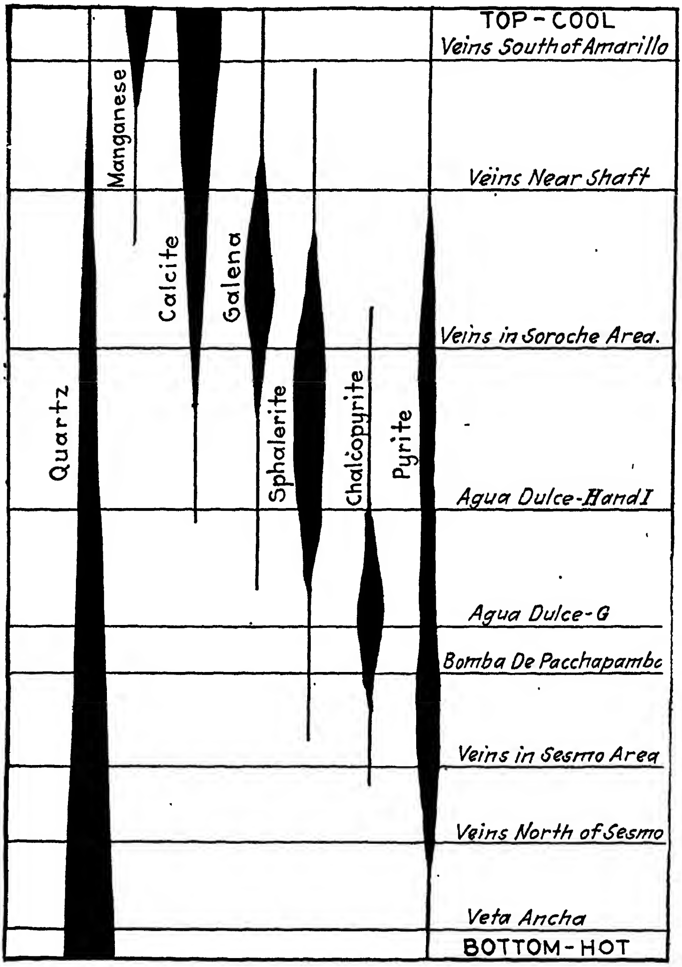


FIG. 5.—DISTRIBUTION OF MINERALS—IDEAL VEIN.

South of the Soroche mine calcite appears as a prominent vein mineral, increasing proportionately until it predominates in the veins near the Portovelo shaft. Still farther southward, across the Amarillo river, manganese occurs with the calcite and quartz. The general distribution of these mineral zones is shown in Figs. 1 and 4.

The principal minerals of these zones are well recognized "vein thermometers." Calcite and manganese carbonate are cool-temperature minerals; chalcopyrite forms at hotter temperatures than sphalerite; and light pyrite occurs in the deep hot portions of a vein. Their relative positions in an idealized vein, hot below and cool above, are shown in Fig. 5.

The horizontal lines on this sketch show the approximate level, or temperature phase, represented by the types of vein filling found in the different parts of the district. The veins showing the hotter phases lie to the north and east, while those with the cool-temperature minerals are found at the southwestern edge of the district. The hottest temperature mineral found is the pyrrhotite, reported by Doctor Colony in the Bosque intrusive, at the edge of the main Amarillo batholith.

The northeastern area is at present from 2000 to 3000 ft. higher than the southwestern. The hot-type vein fillings are now found, therefore, at higher elevations than the cool-type. This is the result of the tilting the region has received. The northeastern area, formerly deep seated, has been raised, and erosion has planed it down until the roots of its veins are exposed at the surface. The southwestern area, relatively depressed, still retains the upper zones of its veins; see Fig. 4.

The relation of the veins to the country rocks emphasizes this condition. North of Zaruma, the veins are found in the very lowest part of the Portovelo formation; even below in the Muluncay series. Southward they occur in successively higher formations until, in the Soroche region, they are at the top of the Portovelo, while Tamayo vein, to the west, extends into the overlying Faique breccias. The lowest levels of the Portovelo shaft are not as deep in the formation as are the vein outcrops in the Sesmo area, north of Zaruma.

The centers with reference to which these temperature zones are oriented are the cupolas formed by the Sesmo and Soroche intrusive stocks. The former has been more deeply eroded and merely the roots of its veins remain, while the latter has been cut down just enough to display the mineral zones to good advantage.

Faulting

As in almost all important mining districts, the adjustment of regional stresses has continued after the period of mineralization, resulting in post-mineral faults. Resulting from very similar causes, this faulting has the same general form and position as the original fissuring. Thus the main southwestern vein system is followed closely by the great Abundancia Fault; the northeastern system is followed by the Tres Reyes-Sesmo Fault; and the two are connected by northeast cross-faults roughly comparable to the northeast connecting veins. However, an important additional set of faults, a northwest series, is found persistently along the

west side of the Abundancia. All the faults dip eastward. Fig. 6 gives the important faults of the region; it may be compared with the vein system as shown on Fig. 3. The present relation of faults and veins to one another is shown in Fig. 7.

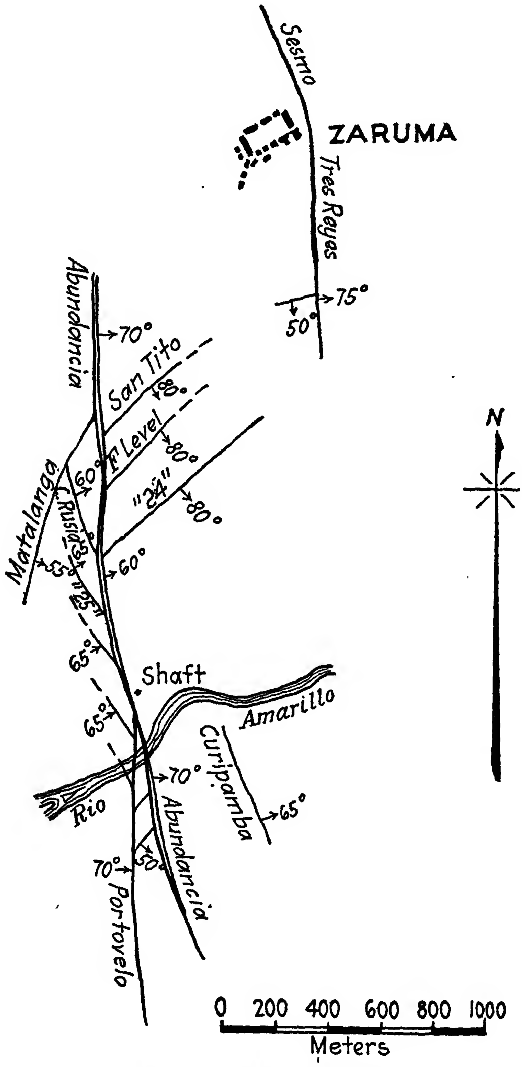


FIG. 6.—FAULT SYSTEM OF ZARUMA DISTRICT.

In most cases, the direction and amount of movement can be determined. Abundancia Fault itself shows strong grooves dipping down to the north at an angle of about 15°. Comparison of the vein segments on

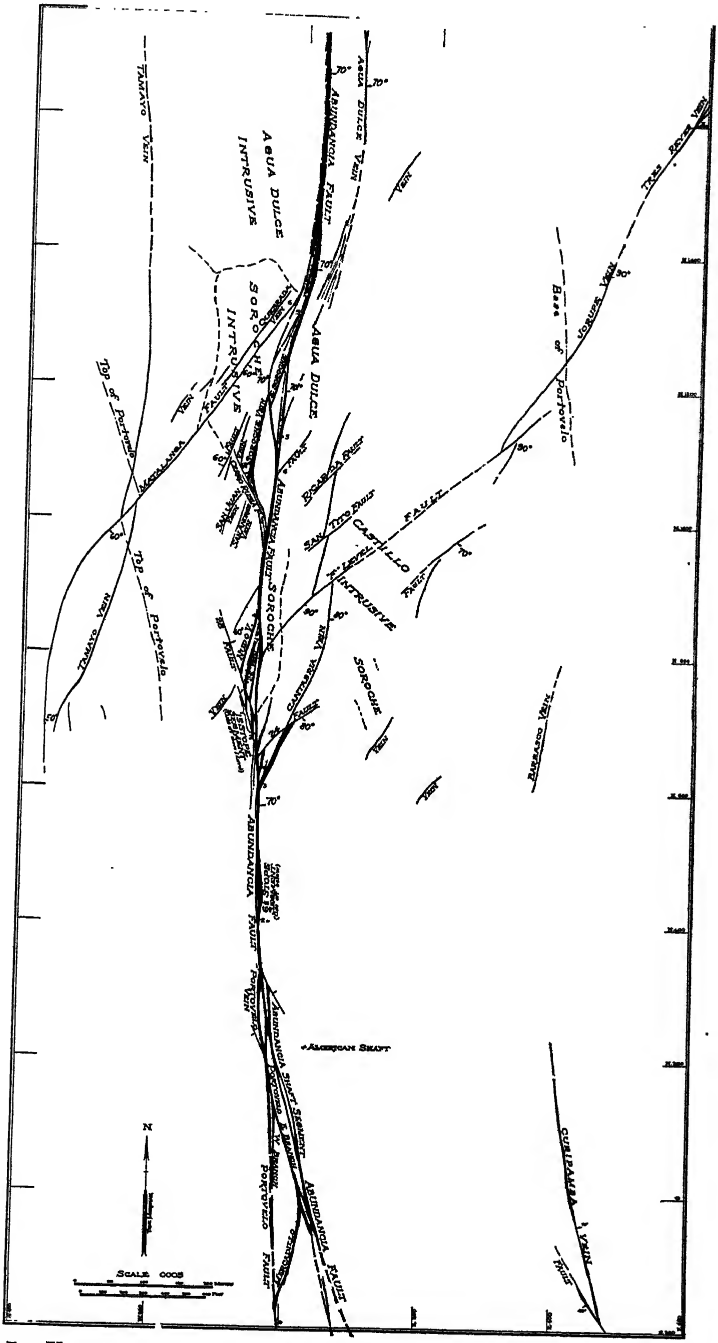


FIG. 7.—VEIN AND FAULT SYSTEM, PROJECTED TO A LEVEL, PORTOVELO MINES.

opposite sides of the fault shows that the eastern side has moved southward and upward along these grooves for a distance of 280 m. (925 ft.).

The other faults are smaller; the north-south faults are always the largest, the northwest faults next in size, and the northeast the smallest. In all, the movement has a large horizontal component, and on the northwest and northeast faults the northern side has always moved toward the west.

The age sequence is as follows: Oldest, northwest; second, northeast, latest, north-south (Abundancia). The faults have nothing to do with the valuable mineralization, unless one is to consider the reopenings of the main Abundancia vein as the first stages of the Abundancia faulting. Whether or not they be so regarded, the greater part of Abundancia movement has been post-mineral. The effect of the faulting has, therefore, been to cut and displace the veins, and to render it more difficult to find and recover all the segments.

Oxidation and Enrichment

During the long period since the tilting of the country, the veins have been exposed to vigorous erosion and oxidation. The amount of secondary enrichment has varied according as oxidation or erosion has predominated. On the steep slopes of the rejuvenated river valleys, erosion has been rapid and has removed the oxidized zone almost as rapidly as formed. Under these conditions primary ore occurs very close to the surface—as in Tres Reyes and Jorupe veins. Where the veins outcrop in the higher hills, which are in part remnants of an older erosion surface, the old deep oxidation may still remain, as in the Gobernadora-Sesmo area. This deeper oxidation has probably caused enrichment of the low-grade primary ores in this region, and has resulted in the shallow rich oreshoots mined in the early days.

The proximity of faults under conditions particularly favorable for water circulation will result in unusually deep oxidation and accompanying enrichment. Matalanga Fault and Tamayo vein provide the best instance.

In general, where neither abnormally rapid erosion, old deep oxidation, nor deep oxidation along faults are present, the veins show a shallow oxidized leached zone, about 10–20 m. deep, below which is a zone of secondary enrichment about 40–50 m. deep. Below this the oreshoots are primary and are distributed in accordance with other factors than distance below the surface.

Development

The rough topography in which the veins of the Zaruma district are found has greatly facilitated their development. The larger veins, with their masses of resistant quartz, have in general become the backbone

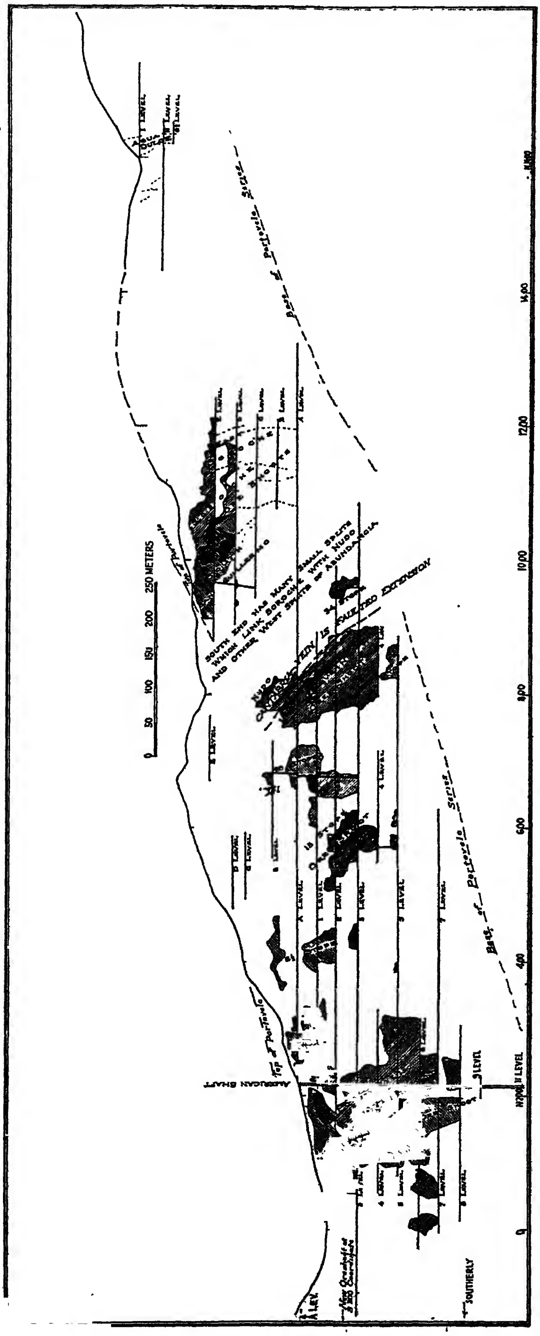
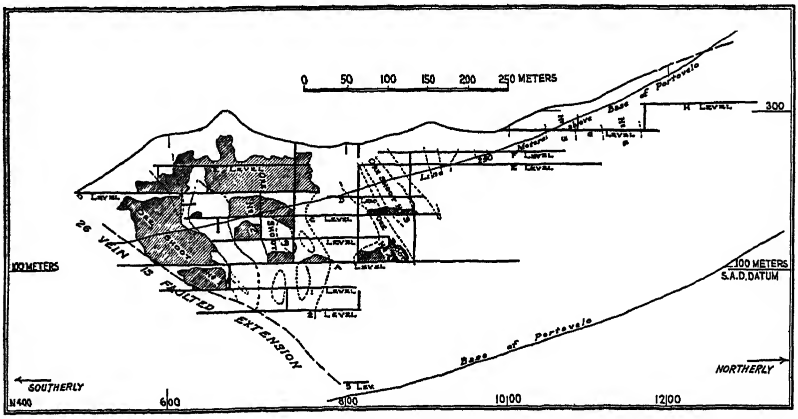


FIG. 8.—LONGITUDINAL SECTION OF ABUNDANCIA AND SOROCHE VEINS, PORTOVELO MINES, LOOKING WESTWARD.

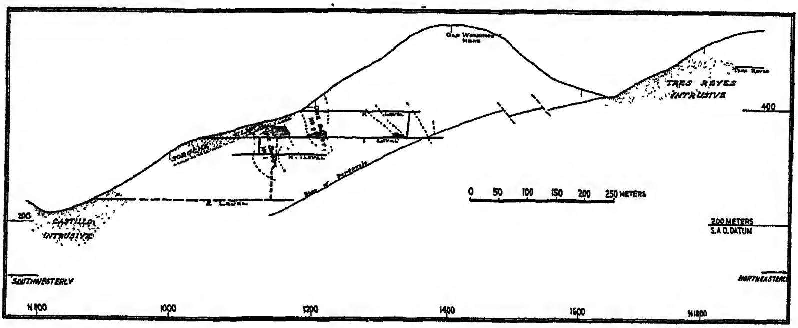
f ridges, between which are sharply incised gulches, or quebradas, from 'hich tunnels can be conveniently driven to tap the veins. The valley f the Amarillo, at the southern edge of the district, affords also a deeply ut trench across the entire vein system.

Five principal veins are now exploited by important workings, in ddition to the numerous smaller veins explored and worked in a lesser



:G. 9.—LONGITUDINAL SECTION OF CANTABRIA VEIN, PORTOVELO MINE, LOOKING WESTWARD.

ay by the Spaniards and their successors. They all lie in the southern ırt of the district, in the vicinity of Portovelo; the mines in the northern 'ea, about Zaruma, being abandoned. These five veins are the Abun- ıncia, Cantabria, Soroche, Jorupe, and Tamayo (see Fig. 3) and the



3. 10.—LONGITUDINAL SECTION OF THE JORUPE-TRES REYES VEIN, PORTOVELO MINES, LOOKING WESTWARD.

erations of the South American Development Co. are essentially nfined to them.

The position of their outcrops, on the northern slopes of the Amarillo lley, has facilitated development by tunnels at many levels, and these supplemented below by nine levels turned from a shaft situated at the

edge of the river bottom. The total vertical range of workings is over 2000 ft. (600 m)., and the horizontal extent is 2500 m. from south to north by 800 m. east and west. Over 50 km. of openings are now accessible.

It would be tedious and profitless to enumerate the levels in detail, but it can be stated that each of these five main veins has been thoroughly developed, except certain fault blocks, from the northern part of the district down to and south of the Amarillo river, where exploration is still in an early stage. Vertically they have been followed, in this area, from the Faique beds, above the Portovelo andesite, down through that formation to its base. Mineralogically speaking, the workings extend from the hot-temperature quartz-pyrite zone, at the northern end,

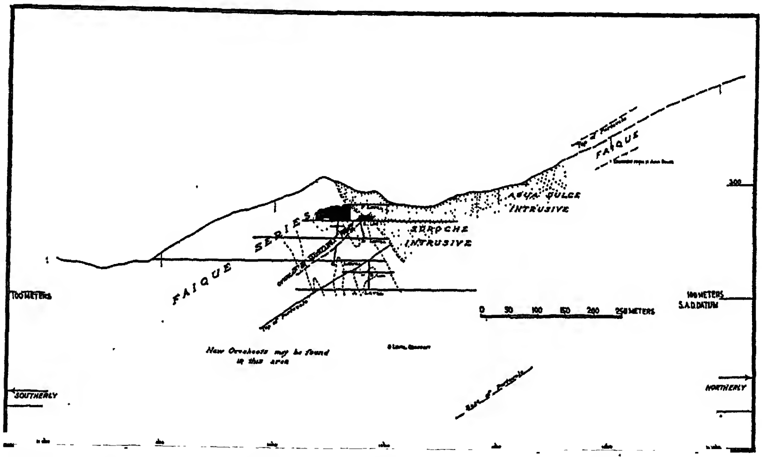


FIG. 11.—LONGITUDINAL SECTION OF THE TAMAYO VEIN, PORTOVELO MINES, LOOKING WESTWARD.

through the intermediate sphalerite and quartz-calcite zones into the low-temperature, calcite-manganese zone at the southern end. The development on each vein is shown in Figs. 8 to 11.

ORESHOOTS

The first impression given by the longitudinal sections is doubtless that the oreshoots are most irregular in shape and position. Some are widest at the surface and diminish rapidly in depth; others do not reach the surface, but appear first underground, increase in size, and then diminish with increased depth; still others show no decrease in size to the depth reached by present development.

If these irregular oreshoots are considered with reference to their position in the formation, however, most of the apparent discrepancies disappear. They are seen, then, to be rather uniform lenses distributed

in the veins along the central and upper portions of the Portovelo formation, getting smaller downward toward the Muluncay and also, probably, getting smaller upward in the Faique, where erosion has not cut below that formation.

These lenses have a constant rake to the north, and the long axis, down this rake, is much greater than the maximum horizontal length of the oreshoot. The true ratio between these dimensions is surprisingly uniform where all the data for its determination are present; that is, where both top and bottom of the oreshoot have been developed, and where the top has not been cut down by erosion. In four such cases the ratio works out to 1.8, 2.0, 1.7, and 2.0, respectively, giving an average of 1.87. In no existing oreshoot does the long axis exceed twice the horizontal length, and seldom is it much less; except where the top has been eroded down or the bottom faulted off. All of the eighteen important oreshoots give evidence of having originally conformed closely to this ratio.

Another feature held in common by the oreshoots is that their center of gravity, so to speak, consistently comes near a certain horizon in the Portovelo formation. This is true both as to size and grade of ore. At and near this horizon—say between 200 and 300 m. above the base of the Portovelo—about one half of the developed length of the vein consists of commercial ore, a proportion that diminishes both above and below these limits.

The commercial ore is not confined to any one zone of mineralization. Even the hottest temperature phases of quartz and pyrite have some pay ore in the parts affected by secondary enrichment, while far out at the cool end old Spanish workings show that ore was found. Most of the oreshoots now being exploited, however, lie in the sphalerite zone of the first generation or in the mixed, quartz-calcite zone of the second generation of mineralization. In the first case, the gold increases with the proportion of sulfides, particularly the sphalerite; in the second, it occurs in bands and spots of dark-colored fine minerals that accompany the calcite generation. Typically, these dark bands are composed of fine-grained pyrite, fine yellow gold, and a third steely gray mineral, rather soft and malleable, with a silvery streak; it tarnishes to a soft black sooty mineral. An assay of the aggregate, carefully hand sorted from the gangue, gave the following results:

Gold	82 oz.
Silver	85 oz.
Iron	much
Copper	0.12 per cent.
Lead	none
Arsenic	none
Antimony	none

It is probable, therefore, that the gray mineral is an alloy of silver and gold, and that the dark sooty tarnish is argentite. In general, throughout the mine, silver is present in the ratio of 2 to 3 oz. per ounce of gold.

Summary

A brief recapitulation of the geological features of the Zaruma district will again emphasize its essential similarity to North American Cordilleran districts. A favorable horizon in the country rock (the Portovelo formation) has been intruded by a series of stocks and sills, some of which, the Soroche-Sesmo intrusives, have been of a type suitable to generate mineral solutions. Fissuring, uplift, and tilting followed the intrusions, and the characteristic linked-vein system received a filling of vein minerals, which arranged themselves in temperature zones outward from their source. Gold accompanied these minerals and was deposited in primary oreshoots in the upper part of the Portovelo formation, particularly in the heavy sphalerite and quartz-calcite zones of mineralization. The adjustment of stresses continued on after the mineralization had ceased, resulting in post-mineral faulting, the movements showing a large horizontal component. The general effect of all the tilting and faulting has been to uplift relatively the northeastern portion of the district, and to depress the southwestern.

The subsequent progress of erosion has tended to reestablish a horizontal surface across the tilted country and has thus worn down more deeply into the northern and eastern parts of the district, so that there only the roots of the veins remain. Under favorable conditions of deep oxidation and enrichment, these have yielded commercial ore, but the bulk of the district's production is coming from the primary oreshoots which, in the southwestern area, still remain below the present surface of the country.

Discussion of Theory of Mine Ventilation

PRESENTED BY A. C. CALLEN,* E. M., M. S., URBANA, ILL.

(New York Meeting, February, 1926)

This report presents the comments of members of the Institute's Sub-Committee on Physics of Mine Ventilation on the proposals of a special committee of the Institute of Mining Engineers (London). This report is as follows:

THEORY OF MINE VENTILATION

Abstract of a report submitted June 16, 1925, to the Council of The Institution of Mining Engineers by a Special Committee

LAWS OF FLOW OF AIR

The law of flow of air in mine galleries is, at present, expressed by two formulas differing slightly in shape but both derived from the original hydraulic formulas of Chezy and others. These formulas are:

The Atkinson Formula.—$Pa = KsV^2$, which in transformation becomes

$$P = \frac{KsQ^2}{a^3},$$

where P = pressure producing flow; K = coefficient of friction; s = surface of the mine galleries; V = velocity of flow; Q = rate of flow; a = area of mine galleries.

This formula is especially applicable to mine galleries, but is unwieldy in application, as in many problems we are dealing with changes produced in the same set of airways when K, s, and a are assumed not to change, and the only variables are P and Q. Great uncertainty prevails as to the exact values to be assigned to the coefficient K in different types of airway, as different workers have given widely differing values. This formula assumes: (1) That resistance to flow is independent of the pressure within the gallery. (2) That changes in volume within the limits of pressure ordinarily met with may be neglected. (3) That changes in density and viscosity and mode of flow of the air may be neglected. (4) That the coefficient K is a constant. (5) That P is proportional to the square of the velocity. We cannot find any reliable experimental work leading us to conclude how far these assumptions are justifiable in mining practice.

Equivalent Orifice Formula.—The fundamental form of this formula is

$$Q = mA\sqrt{2gH}$$

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where m = coefficient of vena contracta; A = area of orifice, in sq. ft.; H = motive column producing flow. In mining practice H is usually written: $A = \frac{0.388Q}{\sqrt{h}}$, where h = in. of water-gage and Q = rate of flow in thousands of cu. ft. per min., in which form a standard air-density has been assumed.

This form (since it was proposed by Murgue) has been used in handling fan problems and, to a lesser extent, in solving problems of flow of air below ground. The principal objections to its use to express resistance are: (1) The coefficient of the "vena contracta" m depends on the size and nature of the orifice. (2) The equivalent orifice is a unit of conductance, and as the direct idea of resistance is usually required, in many problems it becomes necessary to use the equivalent resistance $\frac{1}{A^2}$, thus causing unnecessary complication. Some recent researches on the characteristic curves of fans would be almost impossible to follow if expressed in terms of orifice. An actual orifice might be used as an absolute unit of resistance under strictly defined conditions.

The fact that two equations which cannot easily be expressed in terms of each other are in common use to express the flow of air is confusing, and it would be an advantage if they could be replaced by one simple fundamental form. The inconvenience of both equations for mathematical handling in both research and practical problems.

Conception of Resistance

The idea of resistance as that which opposes motion is commonly used in physical science. Although in hydraulics no unit of resistance has been adopted, its use in electricity is a good example of the convenience and general utility of the conception. In pneumatics, use has been made of the idea in recent work, although no definite unit has so far been adopted.

Influence of Recent Research Work on Laws of Flow of Fluids

In recent years the early work of Osborne Reynolds has been developed by Rayleigh, Stanton and Pannell, and Hodgson and Lacey, and much reliable work, theoretical and experimental, has been carried out on the flow of gas, steam, air and water. To Rayleigh is due the discovery of the principle of dynamical similarity by which it is possible to predict the behavior of any fluid of known density and viscosity from the behavior of any other fluid of known properties, and also the effect of change in dimensions of the duct; on this latter principle is based the use of small-scale models of aeroplanes, ships, etc. These experiments show that the older hydraulic formulas are only approximately true for the following reasons: (*a*) The coefficient of friction is not constant over a wide range of variation of conditions. (*b*) The square law ($P \propto v^2$) is only true in certain

circumstances and for a limited range. The general experience is, however, that a simple index law of the form

$$\text{Resistance of flow} = \text{constant} \times v^n$$

will probably approximate to the conditions found in mining practice. Experimental evidence is, however, urgently required to determine how far such an approximation holds good in actual practice and what the value of the index n should be. The last equation is a general form of which the Atkinson equation is a particular case.

Proposed Restatement of Law of Flow of Air in Mines

It is proposed to adopt the general form originally due to Halbaum and Shaw, and recently restated by Penman, Parker, Briggs and others:

$$P = RQ^n \qquad (1)$$

where P = pressure producing flow; R = resistance to flow offered by fan, mine, or airway; Q = rate of flow.

If experiment shows that the use of the square law is justified in mines, this will be written:

$$P = RQ^2 \qquad (2)$$

The proposed equation $P = RQ^2$ is really a simplified variation of the Atkinson formula, which contains several factors which cannot be accurately ascertained in practice: (1) The constant K, owing to variation in the "roughness" of the surface over which air is passing, also the effect of bends. (2) S, the rubbing surface, cannot be accurately measured in practice for any mine or part of a mine. (3) A, the area, cannot be accurately measured for any mine or part of a mine.

This leaves a simple formula containing three units, two of which can be measured and the third calculated and for a given quantity of air the resistance R is directly proportional to the pressure P. Recent writers and investigators have adopted this form, and it seems desirable to standardize it. Such an equation is equally convenient when applied to fans, mines, or airways. It is easy to express the resistance R in terms of dimensions, etc., of an airway, as from Atkinson's equation:

$$R = \frac{Ks}{a^3}$$

Similarly, the resistance of an orifice can be stated in terms of R, as from the equivalent-orifice formula:

$$R = \frac{W}{m^2A^2 \times 2g}$$

where W = weight of 1 cu. ft. of air.

Conductance

The above expression uses a direct unit of resistance, whereas the equivalent-orifice unit is a unit of conductance. For certain problems it

is desirable to preserve the idea of conductance and to rewrite the equation in the form:

$$Q = C\sqrt{P} \tag{3}$$

where C is a constant representing the conductance of the duct in suitable units. Comparing (2) and (3), we find that

$$C = \frac{1}{\sqrt{R}} \tag{4}$$

The conductance constant will be of service when, say, determining the total resistance of several airways arranged in parallel, for, instead of writing:

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} + \text{etc.}$$

we may write:

$$C = C_1 + C_2 + C_3 + \text{etc.}$$

It is just as easy to preserve the use of two units C and R as it is in electricity where common use is made of the two-fold idea. Moreover, for those mining engineers who have become accustomed to the equivalent-orifice conception it is perhaps less difficult to adopt the suggested treatment, as conductance is proportional to equivalent orifice.

Effect of Leakage

Any statement of the law of flow of air, such, for instance, as the form proposed ($P = RQ^2$) can be true only for a gallery in which there is no leakage. Where there is leakage, the relative effects of friction and of leakage should be separately assessed, and experimental determinations should be made on these lines for different parts of mine-air circuits. The same remark applies to the mine as a whole, and it is unsafe to apply the proposed law, or any other, as a generalization for all mines until more knowledge has been obtained as to the effects of leakage.

Influence of Density of Air

Whereas, in electricity, the resistance of a conductor depends solely on its own properties, in pneumatics, the resistance of a duct will depend on the density (at least) of the fluid, which may change from time to time; therefore the apparent resistance of a mine airway will vary with changes in atmospheric pressure and height above sea level. The best way to overcome this difficulty is to consider density as a function of the resistance, to refer the latter to some standard air density, and to make such adjustment to R and C as may be required in consequence of change of density of the air. The original Chezy formula included a term for density, and expressed in terms of the Chezy formula,

$$R = \frac{Z \times \text{perimeter} \times \text{length}}{2g \times (\text{area})^3} \times W,$$

where Z is the Chezy coefficient of friction. Separating the constants from the variable (density),

$$R = a \times W, \text{ where } a \text{ is a constant;}$$

or, in other words, R will vary directly as the density, and suitable corrections can easily be applied for changes of W. It does not appear to be desirable to disturb the simplicity of the proposed equation:

$$P = RQ^2$$

in view of the above considerations.

Standardization of Units

The great difficulty with the units at present in use is the indiscriminate use of the minute and the second, the foot and the inch, etc., frequently entailing conversion from one to the other in the course of a single problem, and thus leading to unnecessary complication. It is therefore proposed to keep units entirely in terms of the foot, the pound, and the second.

Unit of Rate of Flow

This is at present expressed in three ways:

Cubic feet per second; cubic feet per minute; thousands of cubic feet per minute. It is proposed to adopt the first as this unit is in use in other branches of pneumatics. It is also proposed to give it the name "cusec," as has already been done by irrigation engineers. (Until some form of anemometer capable of giving a direct reading in feet per second comes into general use, routine air measurments should to be made and recorded in cubic feet per minute. For purposes of calculation, it will be necessary to convert the readings to cubic feet per second.)

Unit of Pressure

There are at present three units in use: Inches of water-gage; feet of air-column; pounds per square foot. Of these, the first is unnecessary, being a survival of an old custom only, and invariably entailing the use of a conversion factor. The second, however, is frequently required, but is not an absolute standard, as its value depends on the density of the air; also it cannot be directly measured. It is proposed therefore that the third unit be adopted as the standard and that manometers should be regraduated accordingly.

Unit of Resistance

With the present state of our knowledge we regard with some reserve the suggestion to define an absolute unit of resistance, but for educational purposes, and also for many practical mining purposes, a convenient

standard unit of resistance may be defined as that resistance which absorbs a pressure of 1 lb. per sq. ft. when a volume of 1,000 cu. ft. per sec. of dry air at 60° F. and 30 in. barometer is passing. The name "Atkinson" has been used by recent writers, who have been desirous of recognizing the original work done by J. J. Atkinson, and this name might still be used by those who think it a convenience. The term Kilocusec may conveniently be used to express a rate of flow of 1,000 cu. ft. per sec. It would be possible to construct a circular orifice in a plate that would be of such size that its resistance would be 1 *Atkinson* at standard density of air. The desirability of constructing such a standard orifice, however, is not thought necessary.

The Unit of Conductance

The unit of conductance may be defined as the reciprocal of the square-root of the unit of resistance, and will approximately equal 53 sq. ft. equivalent orifice.

DISCUSSION

OLE SINGSTAD,* New York, N. Y.—The Atkinson formula, $P = K\frac{sV^2}{a}$ was used during the preliminary calculations for the ventilation of the Holland Tunnel, and where the values K, s, and a were constant for a number of cases, it was a simple matter to rewrite the equation in the form $P = K_1V^2$ and avoid the repeated use of identical figures. The Atkinson formula assumes: (1) That resistance to flow is independent of the pressure; (2) changes in volume due to change in pressure may be neglected; (3) changes in density and viscosity may be neglected; (4) that the coefficient K is a constant; and (5) that P is proportional to V^2.

The experiments in connection with the ventilation of the Holland Tunnel showed that all these assumptions could be safely made, except Nos. 3 and 4 at very low velocities. In mine work, where many assumptions must be made, it would seem inconsistent to attempt to make any allowance for the other factors, which are so small that they can be determined only by the most careful experimental work. However, P does depend directly on the density of the air, and to that extent the Atkinson formula is incomplete.

It is suggested that the simple equation $P = RQ^2$ be used, because it is impossible to determine the values of K, s and a in the Atkinson formula. As all these things, as well as air density, affect P, the question arises as to how the value of R would be known for any particular case. It does not seem that the matter is simplified, or at least with equal accuracy, by combining four unknowns into one, especially as at least approxi-

* Chief engineer, The Holland Tunnel.

mate values of the four quantities could be determined for any particular case.

Without doubt, the formula would be convenient after the value of R was known, but this value would depend entirely on the three factors K, s and a, and would be different for each variation in any of the three. If the Committee realized this and suggested the short form for use where the precise values of K, s and a remain the same for a number of determinations, they are suggesting only the thing that would naturally first come to the mind of the computer. If they have in mind the idea of plotting or tabulating values of R in such a way that it would be unnecessary to refer to the Atkinson formula, they have not shown how it is to be done.

In mines already in operation, where the pressure necessary to pass a certain quantity of air is known, the value of R could be computed for further use in the same mine. However, in this case again, the computer would not worry about any formula; he would simply use the square law.

G. E. McElroy* and A. S. Richardson,† Butte, Mont.—A bulletin[1] by the United States Bureau of Mines, contains a somewhat similar discussion of simplified formulas for expressing mine-airway resistances. While it is desirable to have a simple mathematical form for expressing the resistance of mine airways, we do not advocate the entire elimination of use of the Atkinson formula because this formula contains all the laws of ratios required in the simple solution of many mine airway problems and there is no better way of remembering these laws of ratios than by memorizing the formula. Except for the relatively rare cases where R for mine airways can be directly determined, it must be calculated from known data according to the standard formulas or selected from tables or charts using the same data. Character of surface, bends, length, shape and area of airways cannot be eliminated from consideration except in the case of direct determinations of R, which, in the case of mine airways, requires special equipment.

Practically all of the objections raised in the report to the continued use of the standard Atkinson formula apply equally well to the determination and application of values of R, except perhaps that of unwieldiness. To overcome the latter we have advocated and used a modification of the standard formula in which velocity of flow is expressed in thousands of feet per minute and length in hundreds of feet. This change in two of the units results in a change of eight points in the location of the decimal point in K and reduces the figures employed in calculation to very simple proportions.

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* Mining engineer, U. S. Bureau of Mines.

† Ventilation engineer, Anaconda Copper Mining Co.

[1] D. Hamington: Underground Ventilation at Butte. *Bull.* Bur. Mines. (1923).

As a result of our work on mine airways at Butte, we are prepared to say that calculations made according to the Atkinson formula, with allowance for air density, will give results with a greater degree of precision than their application to ordinary rough mine airways. Anyone using the Atkinson formula should know that K is not a constant, but the range of variations in K has been determined with a sufficient degree of accuracy for mine conditions. It is true that, in practice, this formula contains several factors that cannot be accurately ascertained. It is true, also, that the formula does not involve all of the factors that affect mine-airway resistances, but it is our considered opinion, based on much experimental work and acquaintance with mine airways, that the latter factors are of negligible importance in view of the rather large and unavoidable inaccuracies of application of the former.

We do not advocate use of the pounds-per-kilocusec units given in the report under discussion. In mine-ventilation practice in this country, pressures are determined and expressed in inches of water and quantities of flow in cubic feet per minute. Velocity of flow is likewise measured in feet per minute. While these units may not be theoretically sound, they are in almost universal use and for that reason we recommend no change. In order, however, that R may not be too cumbersome a figure for practical everyday use, we suggest the expression of quantity of flow in hundred thousands of cubic feet per minute. With these units, unit resistance is that resistance which required unit pressure—one inch of water—to pass unit quantity of flow—100,000 cu. ft. per min. of air of standard weight, which we take as air weighing 0.075 lb. per cu. ft. As stated in the report, R will vary directly as the weight of air.

Although in the bulletin noted no mention is made of a direct unit of conductance, there are many problems—especially those involving airways in parallel—in which such a unit can be usefully employed, and we therefore favor a unit of conductance. This unit will equal the reciprocal of the square root of R and be determined from the latter for specific problems.

W. S. Weeks,* Berkeley, Cal.—1. The fundamental formula $P = \frac{KsQ^2}{a^3}$ has been sufficiently tested to prove it adequate for ventilation work. Recently many friction factors have been determined, so that a factor for almost any type of duct is available. Many of the fan laws are based on the theory that the pressure varies as the square of the velocity for constant duct, and the tables of the fan manufacturers are computed on this basis because the theory is sufficiently near the truth for engineering practice.

2. There can be no true unit of resistance if the Atkinson formula is used, because the resistance of a duct is a function of a number of vari-

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ables that have nothing to do with the duct. The friction factor K is made up of a factor c and a factor D which is the density of the air ($K = cD$).

The factor c is a variable and changes with velocity, the specific gravity and the viscosity of the air. While it may be possible to ignore the variation of c, the variation in K with change in density cannot be ignored.

3. The substitution of R for a group of purported constants is just a common algebraic device which simplifies certain computations. A large part of the ventilation problem is the prediction of what is going to happen and this entails the use of the complete formula with such values for the friction factors as have been determined. $\frac{1}{\sqrt{R}}$ is Beard's pressure potential of a duct for $\frac{1}{\sqrt{\frac{Ks}{A^3}}} = A\sqrt{\frac{A}{Ks}}$.

This has been in use for many years for calculating natural splitting. The important problem, however, is the one of controlled splitting where different amounts of air are assigned to the different workings and the total mine pressure is computed. The conception of specific resistance does not aid in the solution of this problem.

4. In regard to units: The only advantage of stating flow in kilocusecs is to make the resistance of a duct come out in convenient figures. Air velocity is measured in feet per minute, most of the tables and charts of fan performance and air flow are constructed with cubic feet of air per minute as the unit of quantity. No adequate reason exists for standardizing on a new unit.

The Committee wishes to abolish the inch of water as a unit of pressure. Nearly everyone dealing with ventilation thinks in terms of water-gage. It is used in all tables and curves of the fan manufacturers in the United States and it is used in most of the charts for determining drop in pressure in ducts. In the graphical solution of problems it is the most convenient unit. Many problems require a conversion only at the end and the labor of changing the water-gage to any unit is negligible. A decree would not abolish the water-gage because the fan manufacturers and most of the engineers would pay no attention to it.

In conclusion, if it is desirable to express the resistance of a duct in some unit, the Atkinson is probably as good a unit as any. The effect of density, however, must not be disregarded.

The form $P = RQ^2$ should be looked upon only as an algebraic device to simplify work in certain algebraic solutions. The engineer should be left free to use any of the familiar units that are best suited to a particular solution.

T. Fraser,* Morgantown, W. Va.—There are three new suggestions in the paper that are of interest. The proposed, new, simplified statement of the resistance law $P = RQ^2$ is simply an algebraic expression of the statement that, with other factors constant, the resistance varies as the square of the velocity. This simplified expression will be valuable in teaching work, to impart to the beginning student the fundamental relation of resistance to velocity and may furnish a simpler means of solving some of the conventional classroom problems. In order to make any attempt to estimate the resistance of any given airway system, or even to give the student a complete knowledge of the factors influencing mine resistance, a more complete formula is necessary.

Change from ft. per min. to ft. per sec. for expressing velocity would make mine-ventilation practice conform to the units used in other industries; but, as the practice of stating velocities in ft. per min. and quantities in cu. ft. per min. has been so firmly established here in mining work, fan rating, and especially in the mining laws, it would be difficult to bring the new unit into general use. Such attempted changes have usually been complete failures or have resulted in a confusing double system of units.

Change to ft. per sec. and use of 1000 ft. as the unit of length in Atkinson's formula $P = \frac{KLOV^2}{a}$ eliminates most of the ciphers in the coefficient; or the use of 1000 ft. per min. as the unit of velocity and 1000 ft. as the unit of length does this more effectively. Use of a unit of resistance like the Atkinson is similar except that it does not take into consideration the factors that determine resistance.

Expression of a mine's resistance in Atkinsons gives a definite value for comparison with other mines, regardless of differences in amount of air being circulated. For example, I recently heard of a large mine being developed in southern West Virginia with the preconceived idea of keeping the water-gage down to a certain definite amount per 100,000 cu. ft. of air circulated. Expressing resistance in Atkinsons, I take it to be the plan to keep the airways and ventilation system in such a condition that as the mine grows the resistance, in Atkinsons, will remain constant.

L. W. Huber,† Pittsburgh, Pa.—The report is a step in the right direction to simplify the cumbersome and unscientific mathematics of mine ventilation we now are forced to use. The adopting of a standard unit of resistance for expressing mine resistance should meet with favor.

Our means of expressing mine resistance, in terms of quantity passed at a certain water-gage, is vague and does not permit of ready comparison of the resistances of mines without further analysis. It is conceivable, if

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we used a definite unit of resistance such as the proposed "Atkinson" for expressing mine resistance, that we could, with further extensive experiments, state what the allowable resistances for mines should be, just as electrical engineers have determined what the allowable resistance of good electrical conductors should be. In other words, after we have determined standards, we can much more readily determine, when we know how much air is passing and what the water-gage is, whether our mine is a good conductor of air or not, when we can express the resistance of the mine in units of resistance instead of in the two interdependent expressions, water gage and quantity. The expression "equivalent orifice" was developed for this purpose but, as stated, "equivalent orifice" as expressed in square feet suggests units of conductance rather than resistance. Such expressions as "equivalent orifice," "manometric efficiency," "orifice of passage" and their corresponding mathematical formulas are the reason that the study of mine ventilation and fans is considered difficult and is sidestepped by so many mining men.

It must be borne in mind, however, that the unit of resistance such as the proposed "Atkinson" only simplifies our method of expressing and understanding resistance after we have measured it. We still need the coefficient of friction and the old KsV^2 formula for calculating resistance before we have a current of air flowing. It has been proposed several times in the past that we make up charts or curves for standard sections of entry, standard bends, obstructions, etc., which would show the resistance offered by these entries, bends, and obstructions when different known quantities of air are passing, in inches of water or in pounds per square foot. Heating and ventilating engineers have made up such tables and curves for showing the resistance of galvanized iron ducts used for ventilating buildings and for exhaust systems, but it will be considerably more of a job to compile such charts for mine entries, because of the wide variation in size, cross-sectional area, rubbing surface, and irregularity. However, after the calculations have been made, it would be much simpler to express resistance in definite units and much easier to use the charts in estimating resistance if they are compiled in this way.

In mine-ventilating work in this country quantity is practically always expressed in cubic feet per minute, and this expression is readily changed to cubic feet per second or to thousands of cubic feet per second. If we adopt the Atkinson as the standard unit of resistance and consider it to be that resistance which absorbs a pressure of 1 lb. per sq. ft. when 1000 cu. ft. per sec. are passing it might be advantageous for the sake of uniformity generally to accept the terms "cusec" for cubic feet per second and "kilocusec" for thousands of cubic feet per second as units of rate of flow. This would not be necessary, however, and, if it caused much confusion, would not be advisable at least for everyday mine-ventilating work.

To adopt pounds per square foot as a unit of pressure in preference to the expression "inches of water-gage" would be advisable. Inches of water-gage is a poor and unscientific expression of pressure and it would be a simple matter to calibrate our gages to read in pounds per square foot rather than inches of water, or to change the expression at the time of reading to pounds per square foot if the gage were not calibrated.

While some of the changes in units and expressions may seem confusing at first consideration, I believe their general adoption would result in the ultimate simplification of mine-ventilation mathematics and the study of mine ventilation. If the misconceptions and ignorance expressed by the average mining engineers and operating men today on mine ventilation is any measure of the inadequacy of our present mode of teaching and explaining problems in mine ventilation, we should not have any hesitancy in at least attempting to simplify the mathematics of ventilation.

J. T. Beard, Danbury, Conn.—There are a few basic formulas that point the way to a scientific solution of ventilation problems. We accept the rational formula expressing the velocity v due to a given head h or column of the fluid in question: $v = \sqrt{2gh}$. It appears clear, therefore, that these two prime elements in ventilation bear a fixed relation to each other, the pressure or head varying as the square of the resulting velocity; and it is common to speak of the velocity head, meaning the head generative of such velocity.

The Bernouilli theory applied in determining the hydraulic gradient makes the sum of the velocity head, friction head and gravity head a constant. This principle applies with equal force to conducting air currents in airways. However, owing to the lightness of the flowing medium (air) and the generally level trend of mine airways, the gravity head may be discarded, except in particular cases. The Bernouilli principle establishes the fact that, except, in particular cases, the sum of the velocity head and the resistance or pressure head is a constant, the one increasing as the other decreases, and vice versa.

First, let us understand that, in the steady flow of a medium against resistance, two forces are acting independently and oppositely—the one to produce motion and the other to resist and retard the movement. Evidently, if resistance is nil the rate of motion will be the theoretical velocity of a falling body (*in vacuo*), the entire head being absorbed in producing velocity. On the other hand, if resistance is present a portion of the head is absorbed in overcoming that resistance and the remaining portion then acts to produce velocity under a reduced head. Thus, the actual constant head acts in two parts—one portion acting to remove (eliminate) the resistance and the other to create velocity as under no resistance, each portion acting independently, increasing and decreasing reciprocally, their sum being constant.

In computing ventilation of mines, the outstanding factors in the circulation are the required quantity Q of air and the necessary unit pressure p as determined by conditions in the mine. It is only in comparatively recent years that a ventilating proposition has come to be recognized as incomplete when the desired quantity of air is specified without stating the pressure against which the air is to be circulated. Even today, designers and manufacturers of mine-ventilating fans do not fully comprehend that every mine has a potential value in respect to the passage of air. In other words, the mine has a resisting power dependent on the quantity of air passing, which determines the unit pressure.

Different attempts have been made to designate this resisting power; but whether we call it equivalent orifice, or style it a conductance factor, or otherwise name it, the fact remains that the mine by virtue of conditions within itself determines the ratio of quantity and pressure; the latter varying as the square of the former. I prefer to call this resisting power the mine potential, which seems to describe more fittingly and practically the controlling effect of the mine on the possible circulation of air through its airways. More than three decades ago I drew attention to this factor, in my first book on mine ventilation (1894), and tabulated potential values for different mine airways.

With no intent to disparage the theory of Murgue's equivalent-orifice method of evaluating mines in respect to the circulation of air currents, but to show the fallacy of its unrestricted application in the design and proportionment of mine ventilating fans, I have written an analysis of that ingenious method.[2]

The reserve of the committee in respect to defining a specific unit of resistance, as, for example, that due to the circulation of 1000 cu. ft. per sec., under a pressure of 1 lb. per sq. ft., seems quite proper at this stage of development. Later study of the elements involved in handling and circulating air may suggest a unit based on the work performed in the expansion of a pound of air under a specific pressure—say of one atmosphere or, perhaps, 1 lb. per sq. in. or per sq. ft.

The suggestion of cusec to express cubic feet per second and kilocusec to express a flow of 1000 cu. ft. per sec. are simplifications that readily appeal to the engineer and the statistician. The use of the term Atkinson in the expression of mine resistance does not have the same expressive appeal.

In the study of ventilation, it has been my practice to express the resisting capacity of a mine or airway by the symbol x and call it the mine potential as expressing the relation of quantity and pressure in the circulation of air currents in mines. Best results are obtained by the use of the formula

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[2] *Mines and Minerals* (1896), **17**, 73–76.

$$x^2 = \frac{Q^2}{p} = \frac{a^3}{ks}$$

Whether expressed in terms of the circulation or in terms of the mine, the potential is a factor which evaluates the mine to the extent of determining the possible volume of air that the mine will pass under a given unit pressure. This will appear more clearly when it is remembered that conditions in the mine, alone, determine the relation of quantity and pressure. For that reason, I would substitute the word "potential" for "Atkinson" as more expressive of the meaning of the ratio. The use of a single symbol to express the mine conditions greatly simplifies ventilation formulas.

In discussing the Proposed Restatement of Law of Flow of Air in Mines, I would recommend the acceptance of the square law in relation to quantity and pressure, for reasons previously stated. This will eliminate formula 1.

I would rewrite formula 2, reserving P to express total pressure $P = pa$ producing the air current, and substitute the potential factor x for R, which should be used, as formerly, to indicate mine resistance $R = P = pa$; thus since

$$R = \frac{1}{x^2}, \quad p = \left(\frac{Q}{x}\right)^2$$

In thus using the potential, the formula $R = \frac{K^s}{a^3}$ would be written $x^2 = \frac{a^3}{ks}$. Similarly, the next formula, letting w express the weight of air per cubic foot, reserving W to express the weight of any volume, becomes

$$x = mA\sqrt{\frac{2g}{w}}$$

I would discard the use of the term "conductance" and write formula 3: $Q = x\sqrt{p}$; then formula 4 is not required.

In respect to the summation of two or more airways, the potential enables the ready calculation of a general mine potential (x_0), both in parallel and tandem circulations; and, for split circulations, we write $x_0 = x_1 + x_2 +$ etc. The ratio of the quantity of air passing in any split (q_1, q_2, etc.) to the total circulation Q is then equal to the ratio corresponding split potential x_1, x_2, etc., to the sum of the potentials x_0; thus, $q_1:Q::X_1:X_0$. As this summation of potentials, for both parallel and tandem circulations, is fully explained in Mine Gases and Ventilation (1920) it will be unnecessary to dwell further on this feature.

A. C. Callen,* Urbana, Ill.—The particular comments to be made on this report are on: (1) The use of Q^n or V^n instead of the square of these quantities; (2) the proposed substitution of the formula $P = RQ^n$ for a

* Professor of mining engineering, University of Illinois.

formula of the Atkinson form; (3) the neglecting of density; (4) the change from ordinary units to "coined" units; (5) the use of the equivalent-orifice formula.

1. It seems to be well established, by tests made on ducts or passageways where uniform conditions could be maintained, that pressure does not vary as the second power of the velocity, but at a power somewhat less than two. In spite of this it would appear unwise to break away from the "square law" until more definite data are secured regarding the flow of air in mines and until more accurate methods are devised for the measurement of velocities under ordinary conditions. Unless special air-measuring stations are constructed, there is great question whether average velocities or quantities are determinable with any degree of precision. If errors of 5 or 10 per cent. may be common, it would seem futile to assign different values to n. Let those of us who are working on the problem continue our efforts to clear up this point, and when the true value of n is ascertained, or if it be found to have a value varying with V, there will be no trouble in having it adopted.

Mine ventilation is the hobby of many a mining man as well as a matter of importance to engineers and scientists. Many examinations for certificates of competency require a knowledge of the theory of ventilation, and until we are prepared to give definite values for n it would only complicate matters to insist on a change of thought. No violence will be done to the scientific aspects of ventilation if we continue to follow the "square law" until we are certain that a change is desirable on account of increased knowledge of the subject.

2. The substitution of the formula $P = RQ^2$ for a formula of the Atkinson form is not practicable; the new formula could not possibly replace the old one. It is granted that for certain problems the new formula would be simpler in form, but any one solving problems where K, s and a were constant would of his own accord carry through the value of $\frac{KS}{a^3}$ as a single factor.

The elimination of the Atkinson formula would remove the most important formula in the study of ventilation. There can be no question that much of what has passed for a study of mine ventilation has been simply a course in the mathematical gymnastics of ventilation, and perhaps the Committee feels that a simplification is necessary from this standpoint—but how can the student or engineer appreciate the effect of length, perimeter and area if the new formula is used?

Again, how could the new formula be used in estimating the requirements of a new mine? In the past, too little attention has been given to this phase of the subject, but the time is coming when the estimates for air, fans and power requirements will be prepared with considerable accuracy.

3. If a standardized formula is to be adopted, density is too important to be neglected. An agreement should be reached as to the standard density for which K (or R in the new formula) should always be expressed, and a density factor d, representing the ratio of the density of air at any given temperature and pressure to the standard density, should be added to the formula. The coefficient K includes enough variables and uncertainties without making it carry a factor as definite as density. If anyone desires to neglect variations in density, d becomes unity.

4. The usefulness of any new formula or of new units will have to be judged by use or disuse. Inches of water-gage and cubic feet per minute are so well established wherever air is handled, that it would seem unwise to attempt a change. Furthermore, in certain gages some liquids other than water are used, and in all gages corrections for varying specific gravity at different temperatures must be applied. It would seem that such corrections might be made more readily and with less likelihood of error if applied directly to the original reading in the customary units. This is not a serious matter, but we have always felt that there was an advantage gained by adopting units that had a physical significance. "Three inches of water-gage" has a clearer meaning to most mining men than "15.6 lb. per sq. ft."; the first they can visualize, the second is a derived value. Most men can comprehend feet or thousands of feet (or cubic feet or thousands of cubic feet) per minute better than per second; therefore, while admitting that this is a matter of habit and of law, there would appear to be no good reason for making the change to cusecs or kilocusecs. Such a change would not be generally adopted.

If it seems desirable to employ a formula in addition to, but not as a substitute for, the Atkinson formula it would be desirable to replace the term "resistance" by "specific resistance," "potential," or "inherence." In using the common units, resistance is ordinarily expressed in pounds, and this seems logical. In giving a name to such a unit, there is no question that everyone would agree on the propriety and desirability of calling it an Atkinson.

If a unit of conductance is necessary, it should be given a name.

5. The objections to the use of Murgue's equivalent orifice formula are not apparent. Admitting that the coefficient of *vena contracta* is not correct, the committee has used this same value in calculating a new orifice formula, and it is difficult to see how the new formula is any more simple than the old. It is argued that the equivalent orifice is really a unit of conductance rather than of resistance, but inasmuch as the same formula is used for regulator calculations it would seem that the physical conception of the resistance of an orifice were better established than the conductance concept.

If a new formula is desired, it is suggested that the Atkinson formula be changed to read:

$$i = \frac{dcloV^2}{a}$$

i = inches of water-gage.
d = ratio of density of air at given conditions to density at standard conditions.
c = coefficient of friction or resistance.
l = length in feet.
o = perimeter in feet.
V = velocity in thousands of feet per minute.
a = area in square feet.

The advantages of this form are:

1. It is in the proper shape to use in a wide variety of ventilation problems where the water-gage reading is most convenient to use, and is the logical form because i is the thing actually measured.

2. The numerical value of c is almost exactly (within about one per cent.) the same as the constants obtained when using the metric system and expressing i in millimeters, V in meters per second, o in meters and a in square meters. This simplification would be mutually advantageous.

3. As more attention is given to changes in velocity head and to losses at turns, constrictions, etc., which will be expressed in terms of water-gage, it will become increasingly common to think of resistance in terms of loss of head rather than of pounds or pounds per square inch.

It is true that this would change our method of expressing V from feet per minute to thousands of feet per minute, but it will put us in accord with the English custom.

If a formula for specific resistance is desired, it could be:

$$i = dRQ^2$$

where i is in inches of water-gage, Q in thousands of cubic feet per minute R the specific resistance in Atkinsons and d the density factor. An Atkinson would then be defined as the resistance that would cause a loss of head of 1 in. of water when a volume of 1000 cu. ft. per min. of air of standard density is passing. A circular orifice having an area of about 56 sq. in. (0.388 sq. ft.) would be equivalent to the unit of conductance, and the present orifice formula is unchanged.

This suggestion for a formula for specific resistance is made simply to accord with the suggestions of the committee. The writer is not convinced that such a formula is necessary.

G. S. Rice, Washington, D. C.—A year or two ago, the Ventilation Committee of the British Institution of Mining Engineers

suggested informally that this Institute coöperate in a study of possible standards in mine ventilation. So when copies of this proposed standardization were received, they were sent to members of the Committee. Should we not seek a joint consideration as otherwise we may have different standards in English-speaking countries? In this country, we seem to want merely a modification of existing standards, while some of the British engineers want to start on a new basis.

A. C. Callen, Urbana, Ill.—We should. The Sub-Committee desires a discussion of this matter after which it can formulate its report. The chief objection to this proposed scheme is its tendency to ignore the Atkinson formula so completely and thus wiping out the means of analyzing some of our ventilation conditions.

G. E. McElroy.—It is possible to plot on a chart (such as that shown in Fig. 1) a series of lines denoting the range of friction factors that will be encountered in the application of the Atkinson standard formula for determining mine resistance. It is also possible to plot a series of lines denoting ratio of velocity pressure lost for certain conditions such as bends, etc., that do not involve the characteristics of the airway, such as the roughness of the surfaces, and can be expressed more conveniently in terms of velocity pressures. These losses depend altogether upon the velocity rather than the characteristics of the airway. The chart is based upon the use of the simplified formula; that pressure is equal to the resistance factor times the quantity squared.

In this chart, the resistance factor is denoted by the heavy vertical line that corresponds to the flow of 100,000 cu. ft. per min. The resistance can be determined, for standard conditions, by taking the area of the airway on the bottom scale, then directly opposite the intersection of the proper friction factor line, on the scale on the left is the resistance in inches of water for 100,000 cu. ft. per min. flow through 1000 ft. of airway. For other lengths, the resistance will be in proportion. If the shape varies from the standard, which has been taken as a 1 to 1½ rectangle, certain corrections, which are given on the chart, must be applied. If the density varies from the standard, which is taken as 0.075 lb. per cu. ft., it must be multiplied by the density ratio. Then if the actual resistance at some flow other than 100,000, which is the standard or unit flow, is desired, the resistance factor is first determined and from the proper point on the resistance factor line the parallel to the diagonal ruling is followed to its interception with the vertical ordinate for that many thousands of cubic feet flow, then to the left of the scale the resistance in inches of water for that particular flow may be found.

With a right angle bend, length, of course, is not involved, so the velocity pressure ratio lines—whatever ratio is assumed for the loss for that condition, are used instead of the friction factor lines. For a right

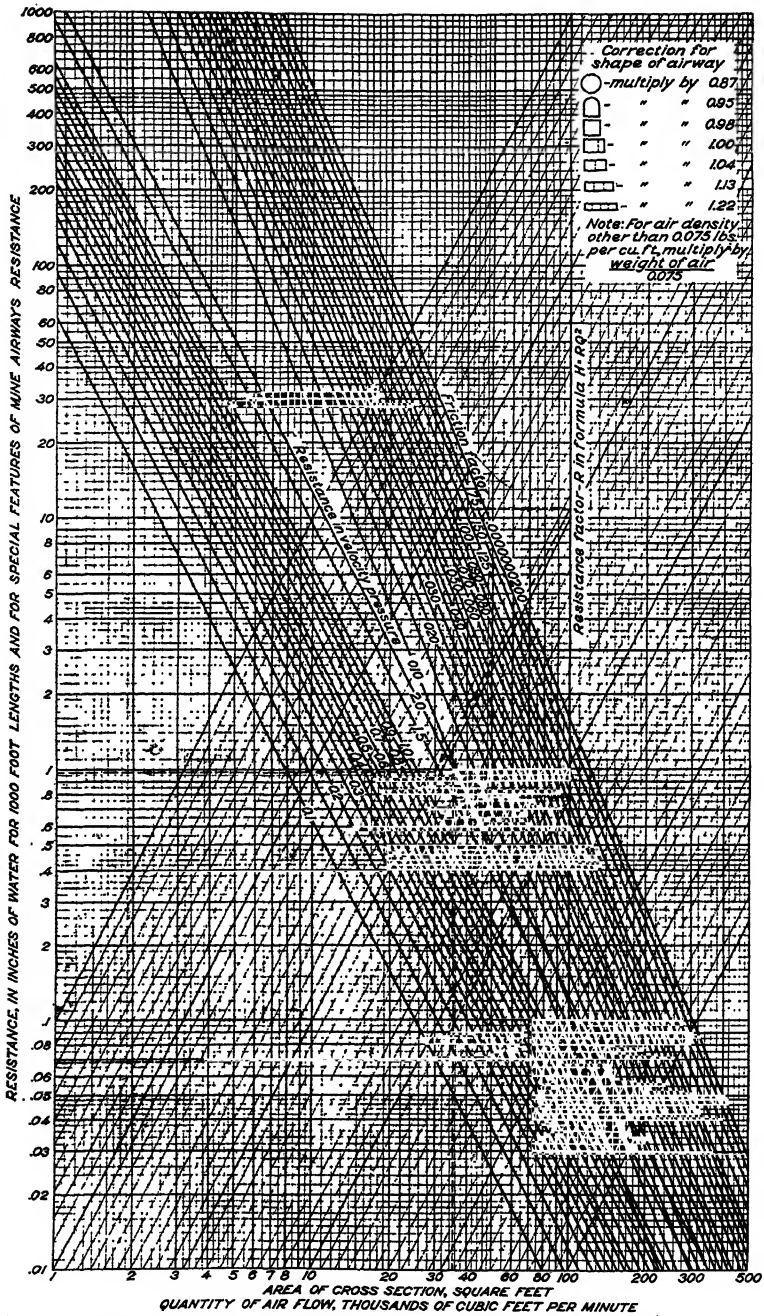

FIG. 1.—CHART FOR DETERMINING RESISTANCE FACTORS* FOR MINE AIRWAYS AND ACTUAL RESISTANCE FOR 1000-FT. LENGTHS OF AIRWAYS AND FOR SPECIAL FEATURES OF AIRWAYS.

* R in the equation $H = RQ^2$, where H is resistance in inches of water, and Q is

angle bend assume a loss of 1½ velocity pressure. Then start with the area of the airway, pass vertically upward to the 1½ velocity pressure line and the intersection gives the resistance for 100,000 cu. ft. flow. If we should have some other quantity, we would proceed from this point on the resistance factor line diagonally downward to its interception with the particular flow ordinate; the resistance is opposite this intersection.

In case the flow is unknown, it is possible to determine the resistance factors for a group of airways and resistance conditions, in terms of 100,000 cu. ft. flow—that is, in unit terms—and combine these for series or parallel flow and arrive at a single resistance factor for a mine or a group of airways; then after determining the factor for the total airway or mine, determine the actual resistance for any particular flow.

B. F. Tillson, Franklin, N. J.—The chart has logarithmic abscissa and ordinates. Under these conditions a straight line upon the chart indicates an equation of exponential form. The natural function of the tangent of the straight line indicates the exponents of the equation. As the two groups of the lines do not show parallel although he refers to the square function; one of those groupings must vary from square function to some exponent.

G. E. McElroy.—With quantity, length, and factor constant, the resistance varies inversely as the cube of the area and directly as the perimeter. Perimeter can be expressed in terms of area and for similar shapes the resistance will vary inversely as the five-halves power of the area. For the standard shape of 1 to 1½ rectangle, the equation of the $0.0^8 10$ friction factor line is

$$Q = \frac{8000}{(\text{area})}\frac{5}{2}$$

from which the equation for any other friction factor line may be obtained by direct ratio of the factor to $0.0^8 10$.

H. L. Dryden, Washington, D. C. (written discussion).—Much of the difficulty with units can be avoided by extending this scheme to the consideration of friction losses and friction factors. Consider the Atkinson formula $P = K\frac{sV^2}{a}$. Calling the perimeter of the section p and the length of duct $l + \frac{s}{a} = \frac{pl}{a}$. But $\frac{a}{p}$ is the familiar hydraulic radius which may be designated by R. Hence $\frac{s}{a} = \frac{l}{R}$. Express P as a ratio to the velocity pressure H equal to $0.00000083V^2d$, if H is in inches of water, d the weight of air in pounds per cubic foot, and V the velocity in ft. per min.

Then $\frac{P}{H} = \frac{KsV^2}{0.00000083V^2ad} = \frac{Kl}{0.00000083dR}$.

Call $\frac{K}{0.00000083d}$ a new factor or coefficient C.

As K in reality was proportional to d, C is independent of d. Then $\frac{P}{H} = C\frac{l}{R}$.

McElroy and Richardson[2] measure P in pounds per square foot and give an average value of K' of 0.000000002 for $d = 0.075$. If P were measured in inches of water, the corresponding K would have been 0.000000000385 and C accordingly 0.0062. If P and H are measured in the same units, inches of water or pounds per square foot, etc., and l and R in the same units, feet or inches, etc., C is independent of the units. Its physical meaning is very definite; it is the drop in pressure expressed as a fraction of the velocity pressure over a distance equal to the hydraulic radius.

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[2] Bur. of Mines Ser. Rpt No. 2540.

Mine-air Flow*

By G. E. McElroy,† Butte, Mont.

(Pittsburgh Meeting, October, 1926)

Much attention has been directed to mine-air flow in recent years, more especially in Great Britian where there is frequent reference to a theory of fluid flow developed by English engineers. Briefly stated, this theory is that for similar passages of the same relative roughness, in which fluid flow occurs, the conditions of flow are exactly the same for equal values of the product $\frac{W D V}{\mu}$, where W = density of the fluid, D = diameter of the passage through which flow occurs, V = mean velocity of flow, and μ = absolute viscosity of the fluid flowing. This, the "Reynolds Criterion" of fluid flow, had its origin in the work of Reynolds[1] more than 40 years ago. Later, Rayleigh[2] amplified the theory which has been conclusively proved for small pipes by the experiments of Stanton and Pannel[3] over a large flow range, and by Lacey[4] over a somewhat more limited range. Concordant results on small pipes have also been obtained by numerous other investigators with single fluids over more or less limited ranges of flow.

According to this theory, results in one medium; such as water, are immediately applicable to another, such as air; and experimental results on models are (theoretically, at least) applicable to full-scale designs. This is of great importance in aeronautics and can be of considerable value in mine ventilation research.

Hodgson, in a recent paper,[5] has outlined the application of this theory to mine-air flow and recent English experiments[6] on the resistance of a timbered mine airway have been expressed in this form.

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* Published by permission of the Director, Bureau of Mines.

† Mining engineer, Bureau of Mines.

[1] *Phil. Trans.* Roy. Soc. (1883), 935.

[2] *Philosophical Magazine* (1899) 48, 321.

[3] *Phil. Trans.* Roy. Soc. (1914) **214A,** 199. T. E. Stanton: Friction (1923) 48.

[4] Stephen Lacey: Flow of Gas in Pipes. *Trans.* Inst. Gas Engrs. (1922–23), 246.

[5] John L. Hodgson: Calculation and Measurement of Air Flows in Mines. *Proc.* South Wales Inst. Min. Engrs. (1925) **41,** No. 4, 417, 554.

[6] Douglas Hay and W. E. Cooke: Underground Tests on Flow of Air at Rockingham Colliery. *Trans.* Inst. Min. Engrs. (1926), **71,** 337.

Application of Theory of Fluid Flow—Experimental Data

The actual results of experiments on mine-air flow so expressed are quite meager, so it is the purpose here to show the relation of data obtained in recent experiments of the Bureau of Mines by this theory of fluid flow; to estimate the importance of the rigid application of this theory to mine-air problems; and to indicate what further experimental data are desirable for a more complete presentation of the matter.

In the accompanying figure (Fig. 1) values of the friction factor K, in units familiar to mining engineers, are plotted against values of the

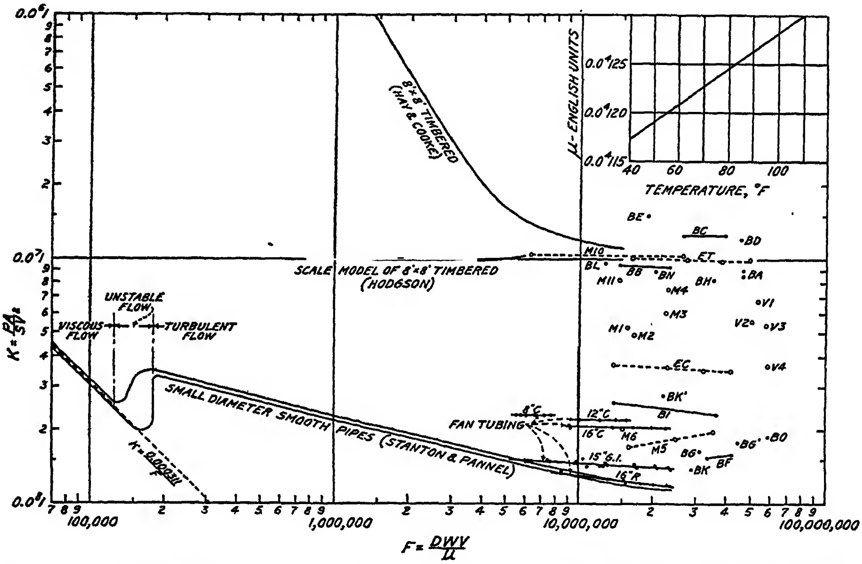

FIG. 1.—DIAGRAM SHOWING VALUES OF FRICTION FACTOR K PLOTTED AGAINST VALUES OF REYNOLDS CRITERION.

[All data are for practically straight airways. See data on facing page.]

Reynolds criterion, also in familiar units. In both cases the units used differ from those by which the subject has been treated previously, which also may be given as a reason for this paper.

K is the "coefficient of friction" or friction factor in the Atkinson formula (after Chezy) for airway resistance: $P = \frac{K\,s\,V^2}{A}$ where P = resistance in lb. per sq. ft. (= 5.2 × resistance in in. of water), s = rubbing surface in sq. ft. (= length × perimeter), A = area in sq. ft., and V = mean velocity of flow, ft. per min. As K includes the density and varies directly as the variation in density, values of K must be related to a standard density, which has been taken as 0.075 lb. per cu. ft.

No.	D	V	
		Fan Tubing—McElroy and Richardson	
8-in. *C*	.64	1760–2490	Heavy canvas, 25 and 50-ft. sections
12-in. *C*	.97	1940–3340	Heavy canvas, 25 and 50-ft. sections
16-in. *C*	1.35	1380–3450	Heavy canvas, 25 and 50-ft. sections
15-in. *G*	1.24	980–3940	Galvanized iron, riveted sections
16-in. *R*	1.35	1250–3520	Rubberized, 25 and 50-ft. sections
		Metal Mine Airways—McElroy and Richardson	
BA	5.7	1700	Timbered crosscut, average
BB	4.3	700–1100	Timbered shaft, 3-comp., average
BC	6.5	800–1200	Igneous rock crosscut, average
BD	9.0	1000	Igneous rock crosscut, average
BE	6.0	600	Igneous rock crosscut, very rough
BF	4.2	1600–2000	Comp. of slab-lined shaft
BG	4.3	1500	Comp. of slab-lined shaft
BG'	6.5	1400	Hand-plastered crosscut
BH	6.8	1100	Timbered crosscut, good
BI	4.6	600–1600	Comp. of solid-cribbed shaft
BK	4.8	1100	Comp. of smooth concrete shaft
BK'	5.0	800	Comp. of rough concrete shaft
BL	5.7	450	Timbered crosscut, average
BN	7.0	600	Igneous rock crosscut, good
BO	6.5	1900	Octagonal solid-cribbed shaft
		Experimental Coal Mine Entries—Greenwald and McElroy	
EC	7.4	300– 900	Very clean and regular
ET	6.4	400–1250	Timbered, average
		Indiana Coal Mine Entries—McElroy	
*V*1	6.9	1200	Very rough floor, regular ribs
*V*2	6.6	1250	Rough floor, regular ribs
*V*3	6.5	1350	Rough floor, regular ribs
*V*4	8.9	970	Clean floor, fairly regular ribs
		Mine Crosscuts—Murgue	
*M*1	6.4	433	Sedimentary rock, very regular
*M*2	6.8	390	Sedimentary rock, very regular
*M*3	7.8	454	Sedimentary rock, irregular
*M*4	4.9	776	Sedimentary rock, very irregular
*M*5	6.5	444–923	Brick-lined, average
*M*6	6.6	371	Brick-lined, good
*M*10	5.8	179–958	Timbered, average
*M*11	6.1	405	Timbered, good

FIG. 1. (*Continued*).—TYPE OF SURFACE.

The Reynolds criterion has been designated by F and its value is $F = \frac{W D V}{\mu}$ where W = density of fluid (air) in lb. per cu. ft., D = diameter of airway in ft. (practically equivalent to four times the hydraulic mean radius, or four times the area divided by the perimeter for other than circular), V = mean velocity of flow, feet per minute, and μ = absolute viscosity in English units. Values of the latter are inserted in the figure as graphed from data given by Hodgson.[7] The idea of air having viscosity may be new to some of us, as this is a more easily conceived attribute of liquids, especially oils, than of gases such as air.

The figure is plotted on logarithmic coordinates, since these are especially applicable to problems of this nature. Horizontal lines or curves indicate resistance varying directly as the square of the velocity,

[7] *Op. cit.*

whereas curves tending downward indicate resistance varying at a lesser rate than the square; 45° downward indicates resistance varying directly as the velocity, *i. e.*, stream-line flow.

The complete range is given by Stanton and Pannel's data[8] for air and water flow through 0.361 to 2.855-cm. smooth brass pipes and shows that fluid flow may be divided into three zones: (1) stream-line or viscous flow at the lowest values of F, (2) unstable throughout a relatively short zone for higher values of F, and (3) turbulent for still higher values of F throughout the more practical ranges of mine-air flow.

The most complete range for mine-air flow and the only experimental data for mine airways not definitely in the turbulent zone is that shown for Hay and Cooke's timbered airway,[9] where velocities as low as 15 ft. per min. were used. For comparison, Hodgson's results[10] on a model of an 8 by 8-ft. airway made to 1/64 scale are also shown. The former was a quite irregular airway, whereas the latter was carefully made very regular.

Data Obtained by Bureau of Mines

Four groups of data, obtained by the Bureau of Mines in typical mine airways, are available: (1) data over practical ranges of F for fan-pipe installations[11] such as are used in the metal mines of Butte, obtained in coöperation with the Anaconda Copper Mining Co.; (2) data for but one or two values of F for a large number of different types of mine airways found in Butte metal mines,[12] also obtained in coöperation with the Anaconda Copper Mining Co.; (3) data for typical coal-mine entries[13] covering a large flow range and obtained at the Bureau's experimental coal mine near Pittsburgh; and (4) data for several coal-mine entries[14] for single values of F obtained through the courtesy of the Knox Consolidated Coal Co. in its American No. 1 mine at Bicknell, Ind.

Graphic Presentation of Selected Data

Selected data from these four groups and a few of Murgue's[15] values for mine airways have been plotted on the accompanying figure, the

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[8] T. E. Stanton: Friction (1923), 53; transposed from Fig. 12.

[9] Calculated and plotted from Fig. 6, page 347, of Hay and Cooke's paper.[6]

[10] Transposed from Fig. 2 of Hodgson's paper.[5]

[11] G. E. McElroy and A. S. Richardson: Friction Factors for Fan-piping Used in Mine Ventilation. Bur. Mines. *Rpts. of Investigation,* (Oct., 1923), No. 2540.

[12] G. E. McElroy and A. S. Richardson: Resistance of Mine Airways. Bur. Mines, *Bull.* 261 (in course of publication).

[13] J. W. Paul, H. P. Greenwald and G. E. McElroy: Ventilation Factors for Coal Mines. Bur. Mines (Bulletin in course of publication).

[14] G. E. McElroy: Unpublished data.

[15] D. Murgue: Experimental Investigation on the Loss of Head of Air Currents in Underground Workings. *Trans.* (1893) **23,** 63.

selection being limited to data for straight and comparatively unobstructed airways. Pertinent data regarding test sections are included in the figure.

From these, and similar graphs, it is apparent that variation from the square relation is greatest for smooth-surfaced airways, and it appears quite probable that above a certain approximate value of K the square relation holds throughout practically all of the turbulent zone; and even for smooth pipe, test results show that this condition is approached as values of F are increased.

The results for 8, 12 and 16-in. canvas pipe show constant values of K for practical values of F, whereas results on rigid piping show downward-trending curves. The difference in results is attributed to the vibration of the canvas pipes, which caused a high degree of turbulence that affected the whole cross-sectional area of flow.

The results for 15-in. galvanized-iron pipe show that the pipe used was relatively somewhat smoother than the pipes of Stanton and Pannel's experiments, although its absolute surface roughness was undoubtedly greater, as it was riveted pipe with slightly coned 6-in. pieces at the end of each 12-ft. mine section to facilitate assembly underground. The lesser slope of the curve is probably due to higher degrees of turbulence introduced by the constrictions. The 16-in. rubberized canvas pipe was an exceptionally smooth line and gives a curve of somewhat steeper slope than the galvanized-iron pipe.

In designating the results of tests on Butte metal-mine airways, the second letter is the original designation of the location at which the test results were obtained. Plotted in this form, errors of determination are easily detected. It is quite apparent that one value of BF must be erroneous and it is quite probable that one value of BB is slightly wrong. These results, as a whole, show the range of factors rather than flow ranges for any particular type of airway.

Coal Mine Results

The experimental coal mine results cover quite a large flow range (300 to 900 ft. per min. velocity in clear entry) and establish the slope of the curves for practical values of F. Actual results were obtained at lower values of F but were so variable that they have not been reported.

The Indiana coal mine results confirm the experimental mine data for clean airways and show average results for typical coal-mine airways in which the floor is covered with slate fallen from the roof.

Murgue's results, for which the numbers indicate his original designation, follow closely the results mentioned except as to $M5$, which is apparently in error. In calculating Murgue's data account was taken of the fact that his anemometers were calibrated on a turntable without

correction for air swirl and he admits that this "excessive registry, which increases with the velocity of the air, can reach 10 per cent. with velocities of 6 m. per sec." His observed velocities have therefore been reduced approximately 1 per cent. per 100 ft. per min. in obtaining these plotted results.

Relative and Absolute Roughness

As has been said, this theory holds for surfaces of *relative* roughness regardless of size. But in practice we deal with differences in size of airways having the same *absolute* roughness and this, I think, is the weak point in the application of this theory to air flow in mine airways. We can determine a friction factor for a particular size of airway having a certain type of surface and from it we wish to estimate the friction factor for a larger or smaller airway having the same absolute roughness of surfaces. Such an extrapolation of data can be made only from basic data showing the variation in relative roughness with change in area for constant absolute roughness. This data is available for diameters up to 30 in. from pipe experiments but is not available for areas comparable with mine airways and can only be roughly computed from the pipe data. The latter indicates that the effect is quite small for the range of areas ordinarily encountered in mine airways. The Bureau of Mines hopes to obtain some data on this subject within the year by determining the resistance of sections of a 400-sq. ft. railroad tunnel with types of surfaces comparable to those of 30 to 50-sq. ft. mine airways upon which determinations have been made.

Division into Flow Zones

Until the appearance of Hay and Cooke's data for low values of F for a timbered airway, it was considered that the division into zones was approximately as shown by Stanton and Pannel's data on smooth pipes and both theory and experimental data based on flow through small pipes indicates that flow in the viscous zone is represented by the line $K = \frac{0.000311}{F}$, so that, in the neighborhood of $K = 0.0^71$ we would expect horizontal lines connected by vertical lines to the viscous flow line at about $F = 150{,}000$. Hay and Cooke's data are therefore contrary to the firmly established theory that viscous flow is independent of both the nature of the surfaces and density of the fluid. The apparent discrepancy could possibly be due to the fact that, in rough airways, a varying degree of turbulence due to projections is imposed on what would otherwise be stream-line flow as it exists in smooth pipes. But Hodgson, in his discussion of Lacey's paper[16] (Plate 5, page 288) gives results by

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[16] *Op. cit.*

Schiller on a coarse-threaded pipe 8 cm. in diameter with a thread 0.8 mm. deep, where the curve agrees with established theory; there is a straight drop from a horizontal line at about $K = 0.0^87$ to the viscous flow line $K = \frac{0.000311}{F}$. Close inspection of Stanton and Pannel's data also indicates the possibility of the offset, or break, from turbulent flow to viscous flow occurring at decreasing values of F as the area of pipe was increased, so that we might expect the turbulent zone for large rough airways to extend practically to intersection with the $K = \frac{0.000311}{F}$ line, that is, beyond the limits of the present graph and away beyond any value of F necessarily considered in mine ventilation.

Application of Hay and Cooke's Data for Low Values of F

With the zone limits as shown by the smooth pipe data (vertical lines), the question of change in factor with change in velocity is of relatively small importance as affecting mine-air flow computations, as the velocities, for which higher factors should be used than those determined at moderate to high velocities, would be so low that they would concern but a very small fraction of the total mine resistance and, considering the necessarily approximate nature of such computations, could be safely neglected. But if these higher values of K should be required for considerably higher values of F, as indicated by Hay and Cooke's curve (and supported to some extent by Hodgson's model experiments), the effect is greater and it is a question whether or not it can be ignored in practical computations.

For smooth airways it would appear that this effect of change of factor with velocity is quite important. And, if a surface is involved the degree of roughness of which is constant and can be absolutely specified, or predetermined, it undoubtedly is of considerable importance. But mine airways are not usually of a fixed and constant degree of roughness and the change in absolute roughness with use or time will often overshadow small changes of factor accompanying change in velocity. For example, consider the two values BK and BK': These were separate compartments of the same concrete-lined shaft but K' had been roughened by falling rock due to skip-hoisting, while K was apparently in its original condition of roughness. Consider also the value BO with a factor of 0.0^819. Previous experiments on this shaft as originally constructed and before any of the timbers had been pushed slightly out of line, gave a factor of about 0.0^815. BI is applicable to exactly the same type of lining but for a different shape and a slightly, but practically imperceptible, greater degree of absolute roughness.

Estimation and Use of Factors

For rough airways, the factor curves are practically straight throughout most of the zone of practical application, say for F equal to 10 to 100 millions, and change of factor with velocity is of negligible importance.

For exceptional cases, it is important to have complete data as to the position and slope of the factor lines and the division into zones of flow. But for general practical purposes—and without very specialized experience it is impossible, ordinarily, to estimate factors closer than 10 per cent.—constant values of friction factors based on determinations in the turbulent zone are sufficiently close and exact for present use. With increase in knowledge and data available, it is quite probable that more rigid general application can be made of such data. In most practical ventilation installations moderate to high velocities are involved. If the use of low velocities of flow becomes more general and such velocities constitute a major part of mine resistances (also where such are the present conditions), it will be necessary to take into account change in factor with change in velocity, or rather, with change in the value of F.

A case in point is the determination of the resistance of mines ventilated by natural draft only. In many cases very low velocities may be involved and large errors are possible in applying data determined at one value of F to flow for the same mine conditions but at much higher values of F.

Reliable determinations throughout as complete flow ranges as can be obtained are desirable for a few typical rough mine airways. Data for very low values of F are not easily obtained, since velocities and pressures are involved that are beyond the range of easily available instruments; and, even with the best of instruments and technique, special conditions are required for success. These determinations should not be attempted on airway lengths less than 1000 ft. with pressure gages reading to 1/10,000 in. of water and velocity determinations should be made at constricted sections. No element of such constrictions should slope at an angle greater than 7°, and the area of the air-measurement section should give velocities not less than 400 to 500 ft. for the lowest test velocity contemplated. In order to decrease the resistance of the air-measurement constriction and thus cover as large a range of flow as the pressure available permits, removable sections might be used so that larger areas at the measurement plane could be used as the flow was increased. Probably the best location for such experiments would be a long, straight, unused split independently ventilated by a fair-sized booster fan. Straight-test sections are preferable for the fundamental determinations, as the laws of variation of pressure loss with change in F for curves and bends should be determined after the data for straight sections is definitely established.

Relation of Velocity Distribution to Slope of Friction Factor Curve

It is quite probable that velocity distributions in uniform-area airways, and especially the relation of the velocity at the center to the mean velocity, are closely allied to the slope of the friction factor curves. Stanton[17] found that the relation of the velocity at any point in a cross-section was independent of the velocity for small pipes when the resistance varied exactly as the square of the velocity (horizontal friction-factor curves) whereas a definite change with change in velocity has been noted by numerous observers in the case of flow in smooth pipes and airways, the rate of change being especially rapid at critical values of F, that is, at the change from viscous to turbulent flow. Fundamental determinations of friction factor curves must be based, therefore, on complete traverses throughout the flow range investigated and this procedure should result in some interesting data on velocity distributions in mine airways.

DISCUSSION

F. E. Brackett, Cumberland, Md. (written discussion).—This paper, based on the "viscosity theory," forms an interesting companion to a paper by the writer[18] in which flow of gases is studied by comparison with a well-known hydraulic formula. As the commonly used hydraulic formulas are based on the "dynamic theory" a discussion of these two theories seems pertinent.

In the subsequent matter, the following notation will be used:

A = area in square feet;
b = a constant;
C = Kutter's coefficient $= \sqrt{\frac{2g}{f}} = \sqrt{\frac{w}{3600k}}$;
c = Hazen-Williams coefficient = C ±;
d = diameter of tube, feet;
f = Fanning's coefficient $= \frac{7200gk}{w}$;
g = acceleration of gravity = 32.2;
$G = \frac{w}{62.4}$;
k = Atkinson's coefficient $= \frac{wf}{7200g}$;
L = length of duct, feet;
P = total friction pressure, lb. per sq. ft.;
p = pressure drop per foot $= \frac{P}{L}$;
R = radius of tube, feet $= \frac{d}{2}$;

[17] *Op. cit.*, 29.

[18] F. Ernest Brackett: The Application of Kutter's Formula to Gases. See page 312.

R_h = hydraulic radius $= \frac{A}{\text{perimeter}} = \frac{d}{4}$;
r = variable radius;
S = fluid slope in duct $= \frac{P}{wL}$;
S_1 = equivalent hydraulic slope $= GS$;
v_x, v_m, v_n, = maximum, minimum, and mean velocities, ft. per sec.;
v = mean or variable velocity, according to contest, ft. per sec.;
V = Mean velocity, ft. per min.;
w = weight of fluid, lb. per cu. ft.;
ϕ_1, ϕ_2 = function of;
μ = viscosity coefficient, English units;
μ' = shear coefficient, English viscosity units.

The dynamic theory is based on the well-known laws of force, mass, and velocity, as applied to a perfect fluid. Among other things, this theory demonstrates that the pressure of a jet against a surface is proportional to the square of the velocity, but it is also affected by the relative angles. The friction of a stream against a surface parallel to its motion is conceived as being due to deflections by minute irregularities in the surface, and therefore is proportional to the square of the velocity. As friction at the sides is balanced by a pressure uniformly distributed over the whole cross-sectional area, shear, or some effect equivalent to it, must exist between the adjacent layers of the stream. In a perfect fluid, this can be accounted for only on the supposition of collision between the atoms, or conflicting currents. Experiments in the main confirm the law of squares, but show a peculiar series of changes in the friction factor, depending on various conditions of the test. The matter is evidently complex both from a mathematical and physical standpoint.

Much light was thrown on fluid flow by Reynolds,[19] who experimented with water flowing in glass tubes, in which "strands" of the stream were colored. He established the fact that there are two kinds of flow, one in which each particle moves parallel to the walls and one where there is a rapid mixing of all the particles; the two systems are separated by the "critical velocity" and its adjacent "unstable zone." Further study developed the fact that fluid flow below the unstable zone followed laws between velocity and pressure that can be established mathematically from the known laws of viscosity; and that the friction factor f is inversely proportional to the Reynolds criterion, $\frac{dwv}{\mu}$. With this fact as a clue, it seemed probable that flow in the "turbulent zone" depended on the same group of factors, so friction factors were plated as ordinates to values of the criterion as abscissas. The results developed a trancendental curve, which is taken as evidence confirming the theory. The equation of this curve is often stated in indefinite form as $f = \phi\left(\frac{dwv}{\mu}\right)$; the exact function

[19] *Op. cit.*

being sometimes considered as exponential and sometimes as a rational binomial.

Thus two theories, based on reasonable grounds, were extended by more or less arbitrary assumptions to the study of turbulent flow, and each, following careful experiments, produced formulas of practical value, although the theories differed fundamentally. The reason for the approximate agreement of results appears to be that both density and viscosity have only slight effect on the value of the friction factors f and C. This can be demonstrated with reasonable accuracy from the exponential hydraulic equations; by deriving a "criterion" similar to the Reynolds, but differing from it in that it does not include the viscosity. For instance, the Hazen-Williams hydraulic formula,[20] $v = 1.318cR_h^{0.63}S^{0.54}$, as shown by Witherell,[21] this can be transformed for any fluid by interpreting S and S_1, the equivalent hydraulic slope, and dividing by the square root of the specific gravity. Then: $v = 1.318c\frac{R_h^{0.63}S_1^{0.54}}{G^{0.5}}$. Treating the Chezy formula in the same manner: $v = C\frac{R_h^{0.5}S_1^{0.5}}{G^{0.5}}$. Solving these two equations for C gives: $C = 1.318cR_h^{0.13}S_1^{0.04}$ and $S_1 = \frac{p}{62.4}$. If the duct is smooth, c may be taken at 140; and the value of Kutter's C, in terms of R_h and p, becomes: $C = 156.4R_h^{0.13}p^{0.04}$. C, and therefore f, seem to be independent of the density when related to the pressure drop p. If C is related to either v or S, instead of p, its value depends to some extent on the density. To produce the value of C in terms of v, it is sufficiently accurate for the present purpose to substitute for p its value derived from the Chezy formula, putting $C = 140$. Fanning's $f = \frac{2g}{C^2}$ is the friction factor generally used in connection with the Reynolds criterion. Its value, found as just indicated, is $f = \frac{0.0058}{R_h^{0.18}v^{0.16}w^{0.08}}$. This equation may be expressed in functional form by affecting the entire right hand member with any exponent, and introducing any new constant factor, if these are convenient. Hence with only slight modification of the individual exponents $f = \phi_2(dvw^{0.5})$. That is for equal values of the product, $dvw^{0.5}$, there will be equal values of f, roughness remaining the same; and this is the use made of the Reynolds criterion, which differs only in containing $\frac{w}{\mu}$ instead of $\sqrt{w}$.

The practical difference between the two theories, then, depends on the difference in value between $\frac{w}{\mu}$ and $\sqrt{w}$; and, as this difference is

[20] Hydraulic Tables: Wiley, N. Y., 1905.

[21] F. Ernest Brackett: Application of Kutter's Formula to Gases, see discussion by C. S. Witherell on p. 321.

affected by a small fractional exponent, its value must be great before its influence on f will be material. In so far as the mobile fluids are concerned, the quantity $\frac{\mu}{\sqrt{w}}$ usually ranges between its value for water, 0.000085, and for air at ten atmospheres pressure, 0.000015. By taking a geometric mean between these two values it is possible to state that:

$$\frac{\mu}{w}\sqrt{w} < 0.000036 \times 2.4 \text{ and } > 0.000036 \div 2.4$$

Hence the factorial error in putting $\sqrt{w} = 0.000036 \frac{w}{\mu}$ is not more than 2.4. But in the exponential formula for f, its value is affected by $w^{0.08} = (\sqrt{w})^{0.16}$; and if the $\sqrt{w}$ or its corresponding factor $\frac{w}{\mu}$ in the Reynolds criterion, contains a factor of error of not more than 2.4, the effect upon f cannot exceed $2.4^{0.16} = 1.15$. Hence whichever theory is assumed as correct, the other is not likely to give values of f more than 15 per cent. in error. (The accuracy of this conclusion is, of course limited by the range of the empirical equations on which it is based.)

To the writer it seems that, at velocities higher than the critical, the principal phenomena are dynamic. The abrupt change in the center constant from 0.5, before critical velocity, to about 1.20, after that point, seems to indicate a sudden "break down" of the fluid; and this seems to be confirmed by the Reynolds experiments. Moreover it can be shown that the apparent shear existing in the stream, with turbulent flow, is, under ordinary conditions, many times the value that can be accounted for by the known viscosity; and its laws differ from the viscosity laws.

The shear, in pounds per square foot, at any cylindrical surface in the stream having a radius r, is evidently $\frac{p\pi r^2}{2\pi r} = \frac{pr}{2}$. Assuming, tentatively, that this shear bears a similar relation to the velocity as that due to viscosity, $\frac{pr}{2} = -\frac{\mu'}{g} \cdot \frac{dv}{dr}$, where μ' is a factor corresponding in units to μ as used in expressing the effects of viscosity; the negative sign indicates that velocity decreases as shear increases.

Recent experiments by the United States Bureau of Mines on ventilating pipe[22] confirm the opinion, heretofore held by many investigators, that the velocity graph under turbulent conditions is parabolic, and that there is a finite velocity of the fluid in contact with the walls. The general equation for the velocity then is $v = v_x - br^2$, where v_x is the maximum or center velocity, b a constant, and v the velocity at radius r. Hence $\frac{dv}{dr} = -2br$; and this varies as the radius. As shear also varies as the

[22] G. E. McElroy and A. S. Richardson: Air Measurement Methods on Fan Pipe. Bur. of Mines, Serial No. 2527 (Sept. 1923).

radius, μ' is a constant in any particular case. In this respect, the law of turbulent shear is similar to that of viscous shear; and this permits a direct comparison between the values of μ' and μ. Putting $-2\,br$ for $\frac{dv}{dr}$ in the tentative equation and solving gives $b = \frac{gp}{4\mu'}$; and the velocity equation becomes $v = v_x - \frac{pgr^2}{4\mu'}$. Let v_n be the velocity at the walls of the tube, and v_m the mean velocity. Then, as the graph is a paraboloid of revolution, $v_m = \frac{(v_x + v_n)}{2}$ and $v_n = v_x - \frac{gpR^2}{4\mu'}$. Hence by substitution, $v_m = v_x - \frac{gpR^2}{8\mu'}$, and

$$\mu' = \frac{gpR^2}{8v_m\left(\frac{v_x}{v_m} - 1\right)}$$

To obtain an expression in terms of Atkinson's K, but $p = \frac{P}{L} = \frac{7200kv_m}{R}$, whence

$$\mu' = \frac{900gKv_mR}{\frac{v_x}{v_m} - 1)}$$

As it has been demonstrated experimentally that for smooth pipes, $\frac{v_x}{v_m}$ is approximately constant at 1.2 over a wide range, μ' is nearly proportional to v_m and R, whereas the viscosity coefficient μ is constant. If the tube is a 16-inch ventilating pipe carrying air at a velocity of 1800 ft. per min., $R = 0.667$, $v_m = 30$. Taking $10^8K = 0.15$ and $\frac{v_x}{v_m} = 1.2$, $\mu' = 0.0043$; and $\mu = 0.0000125$. Hence, in this instance, the turbulent shear is more than 300 times that due to viscosity.

The graph of Hay and Cooke (see Fig. 1) for an 8 ft. by 8 ft. timbered heading is particularly interesting because of the low velocities included in the experiments. Perhaps the unexpected rise of the K line to the left is due mainly to loss of velocity head as the current passes the timber sets and expands to lower velocities between. At high velocities, it is conceivable that the momentum of the current prevents any sensible variation in the general velocity because of this change in area, and that air in the spaces outside of the face line of the timbers is more or less dead; thus making flow similar to an ordinary rough surface. But at very low velocities, the air may undergo a sensible change in velocity inversely proportional to the area. In other words, something similar to a series of miniature cascades occurs and the total loss in pressure is made up of the losses in velocity head at the sets. According to this idea, the following

computation may be made: At $v = 30$, $F = 1{,}500{,}000$ and the graph scales $10^8K = 10$, approximately. At the sets velocity would be $v_1 = 0.5$. Assuming that the area inside the lagging between the sets is 90 sq. ft., the velocity there would be $v_2 = 0.355$; and the loss at each set would be $\frac{v_1^2 - v_2^2}{2g} = 0.00193$ ft. of air, or 0.000145 lb. per sq. ft. Assuming that the sets are 4 ft. apart, this would be the loss in that length of heading; and the value of K, computed in the usual way is

$$10^8K = 10^8\frac{PA}{SV^2} = \frac{10^8 \times 0.000145 \times 64}{128 \times 30^2} = 8.0$$

a figure roughly approximating the graph. In general, so long as this condition lasted

$$P = \frac{0.075v_1^2\left(1 - \frac{v_2^2}{v_1^2}\right)}{2g} = \frac{0.075 \times 0.5V^2}{7200g} 0.00000016V^2$$

and

$$10^8k = \frac{16 \times 64 \times V^2}{128 \times V^2} = 8.0$$

whatever the velocity. Hence, if this theory is admitted, it is necessary also to suppose that cascading effects diminish as the velocity increases.

W. E. Cooke, and D. Hay, Sheffield, Eng. (written discussion).—We have been extremely interested in reading this exposition of the application of the Rayleigh law of dynamical similarity to air flow in mine galleries. While this theory explains the main factors governing air flow, much work must be done before all the phenomena met in practice can be explained. Our experiments at Rockingham have been amplified by experimental observations on a scale model gallery, and also at the new 4-ft. diameter explosion gallery at the Buxton Station of the Safety in Mines Research Board; the results are now in course of publication. We may say, however, that some of the conclusions reached are very similar to those arrived at by the author.

We do not agree that our results are necessarily contrary to the theory that viscous flow is independent of the nature of the surfaces and density of the fluid. The conditions existing in an airway are vastly different from those existing in smooth pipes, and the results obtained can be explained without setting up new theories. For instance, in a large and irregular gallery a wide range of conditions of flow may exist across any given section from extreme turbulence to almost viscous flow; re-entry of air may even occur at certain sections.

We agree that all the experimental evidence available shows that for rough airways the factor lines are practically straight throughout, unless

at velocities far below those normally encountered in mine airways and that for practical purposes the change of factor with velocity is of negligible importance.

We agree, also, that standard determination for typical mine airways through a complete range would be of great service to mining engineers. With the continuation of the practical experimental work on the subject such information should soon be available.

Application of Kutter's Formula to Gases

By F. Ernest Brackett, Cumberland, Md.

(Pittsburgh Meeting, October, 1926)

Much new data on the flow of gases have been discovered by recent experiments by the United States Bureau of Mines and others. Although additional investigation is still desirable, the information now collected seems sufficient to attempt a tentative coördination of the friction factors. This study is confined entirely to fluid friction when the density throughout the duct is so nearly constant that it may be so considered.

In the subsequent discussion the following notation is used:

v = mean velocity of fluid in duct, feet per second.
v' = mean velocity of fluid in duct, meters per second.
R = hydraulic radius of duct, feet.
R' = hydraulic radius of duct, meters.
S = the hydraulic slope.
P = pressure absorbed by friction, pounds per square foot.
p = pressure absorbed by friction, pounds per square inch.
p' = pressure absorbed by friction, kilograms per square centimeter.
A = sectional area of duct, square feet.
O = perimeter of duct, feet.
L = total length of duct, feet.
L' = total length of duct, meters.
d = diameter of a cylindrical duct, inches.
w = density of the fluid, pounds per cubic foot.
g = specific gravity of the fluid compared with water = $w \div 62.4$.
C, k and B, coefficients for formulas using English measures.
C' = metric coefficient, $v' = C'\sqrt{R'S}$.
n = coefficient of roughness for English or metric units.

The basic formula for the flow of all fluids at constant density appears in the three following forms, which may be denominated:

(a) The slope formula, $v = C\sqrt{RS}$

(b) The ventilation formula, $v = \frac{1}{60}\sqrt{\frac{PA}{kOL}}$

(c) The pipe formula, $v = B\sqrt{\frac{pd}{wL}}$

C is Kutter's coefficient, which is extensively applied to the flow of water in ducts and channels of all kinds; k is Atkinson's coefficient, used principally with air in ventilation problems; B is a coefficient conveniently adapted to studying the flow of water, gases and vapors in pipes.

Effect of Density, Radius, and Velocity

By observing that $R = A \div O = d \div 48$, $H = P \div w = 144p \div w$, and $S = H \div L$; the relation between the coefficients is found to be:

$$C = \frac{1}{60}\sqrt{\frac{w}{k}} = \frac{B}{\sqrt{3}}$$

and

$$S \quad \frac{\quad}{C^2R} = \frac{3600\,kv^2}{wR} = \frac{3v^2}{B^2R}$$

With the possible exception of the above relationships, knowledge of fluid friction in ducts is entirely empirical. Numerical values for the coefficients vary greatly with the character of the surfaces in contact, or degree of roughness. Less important variations occur with changes in hydraulic radius and slope, that is, with radius and velocity. Both C and B are "theoretically" independent of the density, which implies that k varies directly as the density. In practice this is found to be nearly exact over the usual range for gases; and under some conditions it is even sufficiently exact to use the same values of C and B for both water and air. However, further investigation shows that the value of C is affected by the density. Variations in the coefficients caused by changes in velocity are greatest at low velocities, becoming less as this increases, until these coefficients become almost independent of velocity. Changes caused by altering the radius are greatest at small radii. With gases, especially the heavier ones, the coefficient is roughly independent of velocity when the latter exceeds about 1000 ft. per min.; and, for pipe, B is often considered roughly independent of size when the diameter exceeds 4 inches.

Application of Kutter's Formula for Water to Gases

Empirical formulas for the flow of gases may be classed under the three following heads: (1) those in which the coefficient is regarded as constant; (2) those that reflect only the influence of the hydraulic radius; and (3) those that give weight to both radius and velocity. All are limited in their range of application, and apply only to surfaces of a stated character. For water, investigations cover a much wider range and the data have been reduced to the formula by Ganguillet and Kutter now known as Kutter's formula. It possesses a practically unlimited range and is universally recognized as a criterion for the flow of water; but it lacks the expression of the effect of density which is necessary for application to gases.

The correlation of the coefficients for gases attempted here is based on the hypothesis that Kutter's formula is correct in form for all fluids (*i. e.* for all densities). It is necessary, then, first to discover the law by

which this formula may be transformed in order to be applicable to other densities; and, second, to compare the results of the transformed equation with existing data. The accuracy of the hypothesis must be judged by the general extent of agreement.

Kutter's formula is:

$$C = \frac{x + \frac{y}{S} + \frac{z}{n}}{1 + \frac{\left(x + \frac{y}{S}\right)n}{\sqrt{R}}}$$

x, y and z are constants; and n is the coefficient of roughness, expressing the character of the surface of the duct. For water, $x = 41.6$, $y = 0.00281$, and $z = 1.811$.

In one important respect the behavior of this formula differs widely from the formulas for gases. Although C is a direct function of R under all conditions of the slope, C is a direct function of the slope (or k an inverse function of the velocity) only when R is less than the square of z. In the formulas for gases, C is a direct function of both R and S under all conditions. This feature of the Kutter formula is illustrated in Table 1, which gives values of C for water abstracted from Trautwine's Handbook.

TABLE 1.—*Values of C from Kutter's Formula for Water* ($n = 0.010$)

$S =$	0.0001	0.001	0.01
$R = 1.00$	147	155	156
$R = 3.28$	181	181	181
$R = 10.00$	205	197	196

Insofar as the writer is aware, this critical radius has never been demonstrated to exist in the flow of gases; but most, if not all, of the data relate to values of R less than 3.28, which is its value for water.

FORMULAS FOR THE FLOW OF GASES

The Babcock Formula for Steam

The Babcock formula for steam[1] is one of the most widely used of the pipe formulas, and may be applied to all gases and vapors. From it

$$B = 265.9$$

This formula is applied to velocities up to 10,000 ft. per minute; but at velocities below 1000 ft., the results are inaccurate and would, in practice, be of little use. Presumably the results apply particularly to wrought iron and steel pipe. Values of B for $d = 1$, and $d = 8$ are 123.9 and 183.4. The range of values given B by a number of representative investigators,

[1] Babcock and Wilcox: Steam. 1902.

who regard this coefficient as constant, varies from $B = 127.1$ (Molesworth) to $B = 172.7$ (Hurst).

The Althans Formula for Gas in Cast-iron Pipe

The Althans formula[2] expresses the flow of gases in cast-iron pipe. By it

$$10^8 k = \frac{0.5833 w^{2/3}}{R^{0.373}}$$

This formula is particularly interesting because many data were examined by its author, and a number of other equations reviewed. According to Althans, good results are obtained with air weighing 0.137 to 0.410 lb. per cu. ft. when the velocity exceeds 591 ft. per minute. But when the density is about 0.078, the results do not agree until velocity is 985. With illuminating gas, when density is 0.031, agreement does not occur unless velocity is in excess of 492. Pipes 2 to 14 in. in diameter were studied, and velocities reached nearly 6000 ft. per minute.

The fractional exponent of w is peculiar as k is generally considered in direct proportion to the density. The explanation of this exponent lies in the general method by which this formula was derived. In effect, the Althans equation is a link between the Devillez experiments on compressed air at high velocities and the Arson experiments on atmospheric air and illuminating gas at low velocities; this link being formed under the assumption that k was independent of the velocity and that the character of the pipe was exactly the same. Thus, prevailing differences in such matters between the two sets of experiments inevitably affect the apparent influence of the density.

The Goodenough Formula for Air in Smooth Concrete Ducts

The Goodenough[3] formula is the most recent, and probably the most carefully investigated, of the ventilation formulas. Transformation of this equation gives

$$10^8 k = \left[1.5115 + \frac{12.382}{v^2 R^2}\right] w$$

Application is to smooth concrete ducts, with air at atmospheric densities. Velocity range is from 1000 to 6000 ft. per min.; and range of radii from $R = 0.20$ to 1.00 approximately.

The close general approximation of the coefficient C for both water and air, together with a clearly defined difference in the laws of variation, is shown in Table 2. Here the water values for C are computed from Kutter's formula by making $n = 0.010$, which is its value for smooth

[2] Prussian Firedamp Commission, 1887.

[3] Based on data secured in ventilation investigation by A. C. Willard for the New York and New Jersey Bridge and Tunnel Commission, 1923.

cement; and these are compared with values for C derived from the Goodenough formula for atmospheric air.

TABLE 2.—*Values of C from Kutter's Formula for Water* (n = 0.01) *Compared with Values Derived from Goodenough's Formula for Air* (w = 0.075)

At constant velocity:				
Velocity;	v =	50.00	50.00	50.00
Radius;	R =	0.20	0.50	1.00
Slope;	S =	0.736	0.276	0.137
Kutter;	C =	116.00	140.00	157.00
Goodenough;	C =	130.00	134.00	135.00
At constant radius:				
Velocity;	v =	16.67	50.00	100.00
Radius;	R =	0.50	0.50	0.50
Slope;	S =	0.034	0.276	1.090
Kutter;	C =	140.00	140.00	140.00
Goodenough	C =	128.00	134.00	135.00

DETERMINATION OF KUTTER CONSTANTS FOR AIR AND GASES

The most obvious way to determine the Kutter constants for air is to assume two radii and two velocities, as widely separated as dependable data on the value of C can be obtained, and to solve the four resulting equations of condition for x, y, z and n. This involves the rather difficult task of selecting four coefficients corresponding to exactly similar surfaces, and as near the extreme scope of definite data as possible. The writer made several trials of this kind, which it is unnecessary to detail here. The results varied considerably according to the assumptions made. Nevertheless these computations led to the following important general conclusions:

1. Value of n for gases with smooth-surfaced ducts is identical with its value for water in contact with similar surfaces.
2. Values of x and z are at least approximately the same for both water and gases.
3. Value of y is the only one of the Kutter constants exhibiting any decisive change; which is that instead of being constant, y is approximately inversely proportional to the density.
4. Hence, to make Kutter's formula suitable for all fluids, whether liquid or gaseous, the only change necessary and warranted by the general data given, is to divide y by the specific gravity (compared with water) of the fluid under consideration. The formula then becomes:

$$C = \frac{41.6 + \frac{0.00281}{gS} + \frac{1.811}{n}}{1 + \frac{\left(41.6 + \frac{0.00281}{gS}\right) n}{\sqrt{R}}} \text{ for English measures,}$$

and

$$C^1 = \frac{23 + \frac{0.00155}{gS} + \frac{1}{n}}{1 + \frac{\left(23 + \frac{0.00155}{gS}\right) n}{\sqrt{R^1}}} \text{ for metric measures.}$$

It will be noted that gS simply takes the place of S; hence, it is only necessary to substitute this value for S in taking C values from published tables or graphs based on Kutter's hydraulic formula. Thereby this popular formula is made applicable to gases and vapors, and very probably also to non-viscous liquids other than water. This linking of the specific gravity with the slope also avoids much inconvenience in expressing loss of pressure (due to flow) per unit length of duct in terms of slope, because;

$$gS = g\,\frac{144p}{62.4gL} = 2.308\,\frac{p}{L} \text{ for English measures,}$$

and

$$gS = g\,\frac{10{,}000p^1}{1000gL^1} = 10\,\frac{p^1}{L^1} \text{ for metric measures.}$$

Hence, when the pressure drop is determined by the conditions of the problem, the equivalent hydraulic slope is easily determined and vice-versa.

Comparison of Results of Kutter's Formula Transformed and Other Formulas

Tables 3 to 8 illustrate the general agreement between the results of the above formula and the usually accepted data pertaining to the flow of gases.

Coefficients for flow in very small ducts at low velocities are out of range of determination by the gas formulas given. In Table 3, some experimental results on pipe are compared with values given by the transformed Kutter's formula. In computing the latter; S is found from values of v and k in the test, and the value of n is adjusted so as to approximately conform to the experimental value of k. These tables are particularly interesting with respect to this value of n, because this quantity will not remain constant for similar surfaces, under variations of slope, radius and density, unless the transformed equation is of correct form. Hence the value of n for smooth ducts is the criterion by which the degree of applicability must be estimated.

TABLE 3.—*Values for Pipes, 3 to 10 In. in Diameter; Velocities Less Than 500 Ft. per Minute*

Kutter's Formula Transformed							Experimental		
S	R	w	n	C	v	10^8k	$60v$	10^8k	Source
0.0520	0.0664	0.031	0.014	31	1.8	0.91	108	0.91	Arson
0.0910	0.0845	0.077	0.013	51	4.5	0.82	270	0.81	Arson
0.0114	0.1020	0.380	0.008	101	3.4	1.04	214	0.97	Devillez
0.0270	0.2050	0.078	0.010	81	6.0	0.33	380	0.30	Arson

At velocities of about 1000 ft. per min., the results from the transformed formula, applied to pipes, begin to check with the Babcock formula, as shown in Table 4. At higher velocities, the agreement is more exact, as shown in Table 5. In both of these n is taken uniformly at 0.009. Slopes are limited to a maximum value of 25 merely to avoid the expression of pressures which, under ordinary conditions, would materially vitiate the assumed condition of constant density.

TABLE 4.—*Values for Pipes, ½ to 12 In. in Diameter; Velocities About 1000 Ft. per Minute*

Kutter's Formula Transformed							Babcock Formula	
S	R	w	n	C	v	B	B	Remarks
9.00	0.010	0.03	0.009	51	15.3	88	93	Coal gas, ½-in. pipe.
9.00	.010	.60	.009	51	15.3	88	93	Steam and compressed air, ½-in. pipe.
2.56	.021	.03	.009	66	15.3	114	124	Coal gas, 1-in. pipe.
2.56	.021	.60	.009	67	15.5	116	124	Steam and compressed air, 1-in. pipe.
.25	.083	.03	.009	88	12.7	152	193	Coal gas, 4-in. pipe.
.25	.083	.60	.009	104	15.0	180	193	Steam and compressed air, 4-in. pipe.
.0625	.250	.03	.009	98	12.3	170	233	Coal gas, 12-in. pipe.
.0625	.250	.60	.009	135	16.9	234	233	Steam and compressed air, 12-in. pipe.

Certain discrepancies exist in Table 4 with respect to coal gas, the difference being greatest with the 12-in. pipe. At this point the transformed equation agrees with the Althans formula, as shown in Table 6, and the range of this last formula seems specifically to cover these conditions (see also Table 7). At the higher velocities given in Table 5, these differences vanish, but a discrepancy exists in Table 6 with coal gas in the 12-in. pipe at $S = 6.25$. Here the Babcock formula is to be preferred; because, when the velocity increases, the value of C for gases tends toward

its value for water, which is tantamount to the friction becoming nearly independent of the density. This tendency, is apparent in Table 2, and is confirmed by a study of Table 8. Value of C given by Kutter for water at $S = 6.25$, $R = 0.25$ and $n = 0.009$ is 139.

TABLE 5.—*Values for Pipes, ½ to 12 In. in Diameter; Velocities* 1500 *to* 10,000 *Ft. per Minute*

Kutter's Formula Transformed							Babcock Formula	
S	R	w	n	C	v	B	B	Remarks
25.00	0.010	0.03	0.009	51	25.5	88	93	Coal gas, ½-in. pipe.
25	.010	.60	.009	51	25.5	88	93	Steam and compressed air, ½-in. pipe.
25	.021	.03	.009	67	48.5	116	124	Coal gas, 1-in. pipe.
25	.021	.60	.009	67	48.5	116	124	Steam and compressed air, 1-in. pipe.
25	.083	.03	.009	105	151.0	182	193	Coal gas, 4-in. pipe.
25	.083	.60	.009	105	151	182	193	Steam and compressed air, 4-in. pipe.
6.25	.250	.03	.009	137	171	238	233	Coal gas, 12-in. pipe.
6.25	.250	.60	.009	139	174	241	233	Steam and compressed air, 12-in. pipe.

TABLE 6.—*Values for Pipes,* 4.32 *and* 12 *In. in Diameter; Velocities* 600 *to* 10,000 *Ft. per Minute*

Kutter's Formula Transformed							Althans Formula	
S	R	w	n	C	v	10^8k	10^8k	Remarks
1.00	0.09	0.030	0.011	77	23.1	0.14	0.14	Coal gas.
4.00	.09	.030	.011	81	48.6	.13	.14	Coal gas.
.16	.09	.075	.010	83	10.0	.30	.26	Atmospheric air.
4.00	.09	.075	.010	93	55.8	.24	.26	Atmospheric air.
1.00	.09	.300	.009	107	32.1	.73	.64	Compressed air.
4.00	.09	.300	.009	107	64.2	.73	.64	Compressed air.
4.00	.09	.300	.008	126	76.0	.52	.64	Compressed air.
0.0625	.25	.030	.009	98	12.3	.09	.09	Coal gas.
6.25	.25	.030	.009	137	171.0	.04	.09	Coal gas.
0.05	.25	.075	.010	97	10.9	.22	.18	Atmospheric air.
1.00	.25	.075	.010	119	59.0	.15	.18	Atmospheric air.
0.0625	.25	.600	.009	135	16.9	.92	.70	Compressed air.
6.25	.25	.600	.009	139	174.0	.86	.70	Compressed air.

As shown in Table 6 a progressive reduction in the value of n is necessary to meet the Althans results, as the density increases. A similar reduction is noticeable in Table 3, where it seems also to separate the Devil-

lez and Arson experiments. These matters are probably closely connected.

TABLE 7.—*Values for Small and Large Smooth Concrete Ducts; Velocities 600 to 6000 Ft. per Minute*

Kutter's Formula Transformed							Goodenough Formula	
S	R	w	n	C	v	10^8k	10^8k	Remarks
0.050	0.250	0.075	0.010	97	10.9	0.22	0.24	Atmospheric air.
1.000	0.250	0.075	0.010	120	60.0	0.14	0.12	Atmospheric air.
0.020	0.500	0.075	0.010	104	10.4	0.19	0.15	Atmospheric air.
0.500	0.500	0.075	0.010	137	68.5	0.11	0.11	Atmospheric air.
0.010	1.000	0.075	0.010	121	12.1	0.14	0.12	Atmospheric air.
0.040	1.000	0.075	0.010	140	28.0	0.11	0.11	Atmospheric air.
0.500	1.000	0.075	0.010	155	110.0	0.09	0.11	Atmospheric air.
0.005	1.500	0.075	0.010	134	11.6	0.12	0.12	Atmospheric air.
0.100	1.500	0.075	0.010	160	62.0	0.08	0.11	Atmospheric air.

The Goodenough formula for concrete ducts has a wider range of radii than the Althans and has the advantage of a greater scope of personally conducted experiments. Table 7 gives some comparisons, the range being slightly extended. Taken as a whole this exhibits a very satisfactory agreement.

TABLE 8.—*Values for Ventilating Pipe and Mine Airways*
[From Experiments by the United States Bureau of Mines (See Serial Nos. 2540 and 2663)]

Kutter's Formula Transformed							Experimental Results		
S	R	w	n	C	v	10^8k	$60v$	10^8k	Experiment on
1.000	0.167	0.075	0.011	96	39.2	0.23	2485	0.232	8-in. ACM canvas pipe.
1.440	0.250	0.075	0.012	95	57.0	0.23	3340	0.220	12-in. ACM canvas pipe.
0.250	0.312	0.075	0.010	121	34.0	0.14	2000	0.142	15-in. galvanized iron pipe.
1.000	0.333	0.075	0.010	128	74.0	0.13	4075	0.120	16-in. rubberized canvas.
1.000	0.312	0.075	0.010	125	70.0	0.13	4500	0.130	15-in. galvanized iron pipe.
1.000	0.333	0.075	0.012	102	59.0	0.20	3450	0.205	16-in. special canvas.
0.500	0.333	0.075	0.012	100	41.0	0.21	2400	0.227	16-in. ACM canvas.
0.003	1.850	0.075	0.019	74	5.5	0.38	300	0.373	Coal mine entry.
0.020	1.850	0.075	0.019	79	15.2	0.34	900	0.351	Coal mine entry.
0.005	1.500	0.075	0.011	121	10.5	0.14		0.10 to 0.20	Smooth-lined shafts.
0.010	1.500	0.075	0.020	67	8.2	0.47		0.30 to 0.70	Sedimentary rock.
0.020	1.500	0.075	0.029	47	8.1	0.94		0.80 to 1.05	Timbered headings.
0.030	1.500	0.075	0.035	39	8.3	1.37		0.90 to 1.95	Igneous rock.

Table 8 gives some of the results of experiments made by the United States Bureau of Mines. Coefficients applicable to large ducts and at low velocities are principally confined to mine ventilation, hence are much affected by the character of the air courses. The values of n applicable to such conditions approximate the values given by Kutter for water in contact with similar rough surfaces.

CONCLUSION

On the whole, the transformed equation seems to agree closely with existing data covering a very wide range. It is desirable, however, that the formula be checked over a much wider range of radii, and that results for low velocities should be extended and determined with greater certainty. Experiments on very light gases would also be interesting, but are probably of secondary importance; for the transformation of applications from water to atmospheric air involves a change of density nearly as 900 to 1, and if the law of density checks through that range it is probably correct for lesser variations. The work given here is merely a preliminary investigation.

DISCUSSION

C. S. WITHERELL, New York, N. Y. (written discussion).—There is no reason to doubt that the relationship between the rate of flow and the impelling force of all fluids can be expressed by a general typical equation that takes cognizance of all relevant factors such as density, viscosity (or its reciprocal, mobility) and nature of channel-walls.

Most fluids that are commonly conveyed by flow in confined channels are so highly mobile that they may be considered non-viscous, hence for these fluids a special factor for viscosity does not need to appear in the equation. Water is the outstanding example of this class of fluids and its dynamic behavior has been so thoroughly investigated that proven formulas pertaining thereto, be they rational or empirical, would naturally serve as prototypes for the other non-viscous fluids. Such formulas take care of "absolute viscosity."

As non-viscous fluids can be classed all gases and vapors, most unsaturated aqueous solutions, gasoline, alcohol, etc. Viscous fluids comprise those that have a sluggish flow such as most oils, tar, molasses, etc., the high viscosity of which is due to shearing resistance between molecules, a force that must be taken into account in figuring the flow of fluids of this class.

The author has convincingly demonstrated in his paper that the same formulas commonly used in hydraulics, with but slight modification, also apply to all non-viscous fluids; in fact, with a different manner of viewing the subject, the transformation is even less than appears in this paper. The author uses S as the hydraulic slope, figured in the same way for gases as it is for water. S is simply a ratio $\frac{h}{l}$, where h is expressed in the same units as l. But for gases in conduits h must be obtained from $\frac{P}{w}$, where P and w are in the same units as l; for example, if l is in feet, P is in lb. per sq. ft. and w is in lb. per cu. ft. For water

w then has the value 62.4 but for air w is only about 0.08 for 0° C., 760 mm., and standard gravity, and hence S is quite large.

But if the drop in pressure, P, per unit of length due to flow is measured with a water manometer column and h so ascertained is expressed in the same unit of length as l, the value of S_1 so obtained is not the gas slope but the water slope equivalent to the gas slope, and would be the same as the quantity gS in the author's transformation of Kutter's formula for finding the coefficient C. Hence if he had used S_1 the equivalent hydraulic slope in place of S, the specific gravity g, would not need to appear and the formula could retain its original form. As S_1 has been substituted for gS in the computation for C, it must be substituted in the formula itself, and as $S = \frac{S_1}{g}$, in order for the usual slope formula, $v = C\sqrt{RS}$, to be applicable to a fluid of any density it should be written $v = C\sqrt{\frac{RS_1}{g}}$, where S_1 is the hydraulic slope equivalent to the gas slope, measured as explained above.

Flow-coefficient Tables

Impressed by the soundness of this demonstration I prepared the two tables given here. As C can not be determined until some value depending on C is first approximately determined, such tables eliminate a long tedious guess-and-try procedure. In both tables the quantity S is as the author defines it.

The problems pertaining to pipes or other circular conduits are usually of the nature of determining the diameter for conveying a given quantity of fluid with a specified loss of pressure due to flow. The table entitled "Flow Coefficients for Circular Conduits" was compiled with this feature in view. The quantity Q may be defined as the equivalent weight of water flowing per second that causes the same loss of pressure as the given weight of the fluid in question. As Q increases in value as d increases, the cube roots are tabulated in preference in order to facilitate interpolation. The coefficient of surface roughness (n) was arbitrarily taken at 0.012 as being representative and safe for materials commonly used.

Ventilation problems, particularly of mines, are usually of the nature of determining the pressure necessary for moving a given quantity of air through given sizes of passage. Given the volume of air flowing per unit of time and the transverse area of passage the velocity is determined. The table entitled "Flow Coefficients for Air Passages" was compiled with this feature in view. The surface coefficient n is a matter of judgment as in hydraulics. Probably the value $n = 0.040$ is the ultimate; $n = 0.035$ is given for igneous rock in Table 8. It will be noted that the air passage table is calculated for air at 70° F. and at sea level. As the ventilating efficiency is a function of weight rather than volume of air

moved per unit of time, it is best to first solve such problems on the basis of 70° F. and sea level and then correct result found for actual temperature and altitude.

The meaning and method of using either table is made clear by superscript and examples. In practice, it would seldom be necessary to resort to such close interpolations as given in the examples; mere mental interpolations would most times suffice.

It is to be understood that the loss of pressure due to flow (mathematically expressed by $\frac{\partial P}{\partial L}$) does not include resistance of bends and other miscellaneous obstacles.

Examples of Flow Calculations Illustrating Use of Flow Coefficients for Circular Conduits Table

Example C.1.—Required the diameter of a ventilator pipe handling 500 cu. ft. per sec. at a friction loss of ½-in. water-pressure per 1000 ft. Sp. g. of air at sea level at 70° F. = 0.0012, hence $\sqrt{g} = 0.03464$, and $W = 500 \times 0.0012 \times 62.4 = 37.44$ lb. per sec.; $Q = 37.44 \div 0.03464 = 1081$; $Q^{1/3} = 10.26$, and $p_L = \frac{0.5}{12 \times 2.308 \times 1000} = 0.000018$.

Since neither the value $Q^{1/3} = 10.26$, nor $p_L = 0.000018$, can be taken directly from the table Flow Coefficients for Circular Conduits, it is necessary to interpolate between the values 9.09 and 10.78 for $Q^{1/3}$, which appear in the column headed $p_L = 0.000001$ in the table. This gives the value $d = 68.31$. Similarly interpolating in the column headed $p_L = 0.00003$ gives $d = 54.41$. Next interpolating between these two columns, using for this purpose the log p_L and the values of d tentatively determined above, gives the final tabular value for d of 60.87. Using this value for d, we next interpolate in the column headed $p_L = 0.000001$ in the table for c, obtaining the value 25.14. Interpolating in the column headed $p_L = 0.00003$ gives the value $c = 27.24$ and interpolating between these two columns as before gives the final value $c = 26.26$. This result may be checked by computing the value of d from the equation $\log d = 0.4(\log 10.81 - \log 26.26) - 0.2 \log 0.000018 + 0.181723$. From this, $d = 60.53$, which is a fair check with the interpolated tabular value 60.87.

Example C. 2. a.—Required the diameter of a furnace blast-main to deliver 3000 tons of air per day at 32 oz. pressure. Altitude 10,000 ft. (Barometric pressure = 10 lb. per sq. in.) Allowable loss of pressure 0.5 oz. per 100 ft.

Sp. g. at 70° F. $= \frac{10 + 2}{14.7} \times 0.0012 = 0.000980$, and

$\sqrt{g} = 0.03130$.

$W = (3000 \times 2000) \div (60 \times 60 \times 24) = 69.44$ lb. per sec.

2	1.414	3.730	0	4.805	.2521	6.499	.3408	8.765	.4521	10.79	.5922	11.70	.7307	12.08	.9026	12.19	1.087	12.23	1.331	12.24	1.598	12.25	1.954
3	1.732	4.568	0	5.837	.3771	7.794	.5076	10.34	.6697	12.53	.8728	13.50	1.075	13.90	1.326	14.02	1.597	14.07	1.954	14.08	2.348	14.08	2.870
4	2.000	5.275	0	6.694	.5017	8.845	.6729	11.57	.8838	13.87	1.147	14.87	1.410	15.28	1.739	15.40	2.095	15.45	2.562	15.46	3.078	15.46	3.762
5	2.236	5.898	0	7.440	.6258	9.742	.8369	12.60	1.095	14.96	1.417	15.98	1.740	16.39	2.144	16.51	2.582	16.56	3.158	16.57	3.794	16.57	4.637
6	2.449	6.460	0	8.104	.7496	10.53	.9998	13.49	1.304	15.89	1.683	16.90	2.064	17.32	2.542	17.44	3.060	17.48	3.744	17.49	4.497	17.50	5.497
8	2.828	7.460	0	9.271	.9964	11.88	1.323	14.96	1.716	17.39	2.205	18.40	2.698	18.81	3.321	18.93	3.997	18.97	4.889	18.98	5.873	18.99	7.178
10	3.162	8.341	0	10.28	1.242	13.01	1.642	16.17	2.120	18.59	2.715	19.59	3.318	19.98	4.082	20.10	4.912	20.14	6.007	20.15	7.216	20.16	8.820
12	3.464	9.137	0	11.18	1.487	14.00	1.959	17.19	2.519	19.59	3.216	20.57	3.925	20.95	4.827	21.06	5.808	21.11	7.103	21.12	8.532	21.12	10.43
14	3.742	9.869	0	11.99	1.731	14.88	2.273	18.08	2.913	20.45	3.709	21.40	4.522	21.77	5.559	21.88	6.688	21.92	8.179	21.93	9.824	21.93	12.01
16	4.000	10.55	0	12.74	1.974	15.68	2.585	18.87	3.303	21.19	4.196	22.11	5.111	22.48	6.280	22.58	7.554	22.62	9.238	22.63	11.10	22.64	13.56
18	4.243	11.19	0	13.43	2.216	16.40	2.895	19.58	3.688	21.85	4.676	22.75	5.691	23.10	6.991	23.20	8.408	23.24	10.28	23.25	12.35	23.25	15.10
20	4.472	11.80	0	14.08	2.458	17.07	3.203	20.22	4.070	22.44	5.151	23.31	6.265	23.65	7.693	23.75	9.253	23.79	11.31	23.80	13.57	23.80	16.61
24	4.899	12.92	0	15.27	2.940	18.27	3.814	21.34	4.824	23.47	6.086	24.29	7.393	24.60	9.074	24.70	10.91	24.73	13.34	24.74	16.03	24.74	19.59
28	5.292	13.96	0	16.34	3.419	19.32	4.418	22.31	5.567	24.33	7.003	25.10	8.498	25.39	10.43	25.48	12.54	25.51	15.33	25.52	18.41	25.52	22.50
32	5.657	14.92	0	17.32	3.896	20.26	5.018	23.15	6.299	25.07	7.906	25.79	9.586	26.07	11.76	26.15	14.14	26.18	17.28	26.19	20.76	26.19	25.37
36	6.000	15.82	0	18.22	4.371	21.11	5.612	23.90	7.023	25.72	8.796	26.40	10.66	26.66	13.07	26.74	15.71	26.77	19.21	26.77	23.07	26.77	28.19
42	6.481	17.09	0	19.47	5.081	22.26	6.494	24.88	8.094	26.56	10.11	27.18	12.24	27.42	15.00	27.49	18.03	27.51	22.04	27.52	26.47	27.52	32.35
48	6.928	18.27	0	20.60	5.787	23.28	7.367	25.73	9.149	27.28	11.40	27.84	13.79	28.06	16.89	28.12	20.30	28.15	24.82	28.15	29.81	28.15	36.44
54	7.348	19.38	0	21.64	6.490	24.20	8.233	26.48	10.19	27.90	12.67	28.42	15.31	28.62	18.76	28.68	22.54	28.70	27.56	28.70	33.10	28.70	40.45
60	7.746	20.43	0	22.61	7.190	25.03	9.090	27.16	11.22	28.46	13.93	28.93	16.82	29.10	20.59	29.16	24.75	29.18	30.25	29.18	36.33	29.18	44.41
72	8.485	22.38	0	24.37	8.581	26.50	10.78	28.31	13.24	29.39	16.39	29.78	19.77	29.93	24.20	29.97	29.07	29.98	35.54	29.99	42.68	29.99	52.17
84	9.165	24.17	0	25.94	9.963	27.77	12.46	29.28	15.23	30.17	18.80	30.48	22.65	30.60	27.72	30.63	33.30	30.64	40.70	30.64	48.88	30.65	59.74
96	9.798	25.84	0	27.35	11.33	28.88	14.11	30.11	17.18	30.82	21.16	31.07	25.48	31.16	31.17	31.19	37.44	31.19	45.77	31.20	54.96	31.20	67.18
108	10.39	27.41	0	28.63	12.70	29.89	15.74	30.84	19.10	31.38	23.48	31.57	28.26	31.64	34.56	31.66	41.51	31.67	50.74	31.67	60.94	31.67	74.48
120	10.95	28.89	0	29.83	14.05	30.76	17.35	31.48	21.00	31.87	25.77	32.01	30.99	32.06	37.90	32.08	45.52	32.08	55.64	32.08	66.82	32.08	81.67
144	12.00	31.65	0	32.00	16.74	32.33	20.53	32.57	24.72	32.70	30.25	32.74	36.35	32.76	44.44	32.76	53.37	32.77	65.23	32.77	78.33	32.77	95.74
180	13.42	35.39	0	34.78	20.73	34.24	25.21	33.86	30.16	33.66	36.79	33.60	44.15	33.57	53.95	33.57	64.79	33.56	79.18	33.56	95.09	33.56	116.2
240	15.49	40.86	0	38.58	27.28	36.72	32.80	35.47	38.93	34.83	47.29	34.62	56.68	34.54	69.23	34.52	83.12	34.51	101.6	34.51	122.0	34.51	149.1
300	17.32	45.68	0	41.68	33.71	38.62	40.17	36.65	47.40	35.67	57.41	35.35	68.74	35.24	83.93	35.20	100.8	35.19	123.1	35.19	147.9	35.19	180.7
360	18.97	50.04	0	44.32	40.05	40.16	47.37	37.58	55.64	36.32	67.24	35.90	80.43	35.77	98.19	35.73	117.9	35.71	144.0	35.71	173.0	35.70	211.4

Approximately correct values will be obtained by direct interpolation but greater accuracy is obtainable by the following procedure.
When interpolating for c between d values, or for d between c values, use $\sqrt[4]{d}$ values; particularly for low values of d.
When interpolating for c, d or $Q^{\frac{1}{3}}$ between p_c values, or for p_c between c, d or $Q^{\frac{1}{3}}$ values, use $\log p_c$ values; except between $p_c = 0$ and .000003.
When p_c is less than .000003 then $Q^{\frac{1}{3}}$ varies approximately as $\sqrt[3]{p_c}$.
For p_c values greater than .1 there are practically no corresponding changes in the values of c.

Compiled by C. S. Witherell
June, 1926

FLOW-COEFFICIENTS *for* CIRCULAR CONDUITS

BASED ON KUTTER'S FORMULA TRANSFORMED BY F.E.BRACKETT

Applicable to any non-viscous fluid, whether a gas, vapor or liquid.

Basic formulas:—

$v = C\sqrt{RS}$ ft. per sec.

$$C = \frac{41.6 + \frac{.00281}{gS} + \frac{1.811}{n}}{1 + \frac{\left(41.6 + \frac{.00281}{gS}\right)n}{\sqrt{R}}}$$ for English measure.

wherein:—

v = average velocity (ft. per sec.) throughout the transverse section.
C = Kutter's coefficient (transformed by Brackett).
R = transverse area (sq.ft.) divided by perimeter (ft.).
S = slope of fluid-gradient, figured in same manner as hydraulic-slope.
g = specific gravity of fluid compared with water.
n = coefficient of surface roughness, which is equal to .012 for ordinary iron, unplaned lumber, reasonably smooth cement, etc., and is the value assumed for this table.
It is possible for n to be as low as .009, e.g. for especially smooth metal, glazed even surfaces, etc.; in which case d is reduced about 12% when p and Q remain constant.

Derived formulas:—

Let p = loss of pressure due to flow, expressed in lbs. per sq. inch per ft.

then $S = \frac{144p}{62.4g} = 2.308\frac{p}{g}$, and $gS = 2.308p$

Let d = diameter of conduit in inches

then $R = \frac{d}{48}$ and $C\sqrt{RS} = 0.2193\,C\sqrt{\frac{dp}{g}} = c\sqrt{\frac{dp}{g}} = v$ ft. per sec.

Hence from the foregoing:—

$$c = 0.2193\,C = \frac{42.21 + \frac{.000267}{p}}{1 + \frac{3.459 + \frac{.0001012}{p}}{\sqrt{d}}}$$

Volume, cu.ft. per sec., $V = \frac{\pi d^2 v}{4 \times 144} = \frac{.005454\,c\sqrt{d^5 p}}{\sqrt{g}}$

Weight, lbs. per sec., $W = 62.4\,g\,V = \sqrt{g}\,(.3403\,c\sqrt{d^5 p}) = Q\sqrt{g}$

hence $Q = \frac{W}{\sqrt{g}}$ and $Q^{\frac{1}{3}} = \sqrt[3]{\frac{W}{\sqrt{g}}}$, the "cue" for approximating c, d or p.

$\text{Log}\, d = 0.4(\log Q - \log c) - 0.2\log p + 0.18723$

$\text{Log}\, p = 2(\log Q - \log c) - 5\log d + 0.93617$

Actual diam. d	$\sqrt{d}$	$p=0$, $\sqrt{p}=0$, $\log p=-\infty$		$p=.000003$, $\sqrt{p}=.001732$, $\log p=\bar{6}.47712$		$p=.00001$, $\sqrt{p}=.003162$, $\log p=\bar{5}.00000$		$p=.00003$, $\sqrt{p}=.005477$, $\log p=\bar{5}.47712$		$p=.0001$, $\sqrt{p}=.01$, $\log p=\bar{4}.00000$		$p=.0003$, $\sqrt{p}=.01732$, $\log p=\bar{4}.47712$		$p=.001$, $\sqrt{p}=.03162$, $\log p=\bar{3}.00000$		$p=.003$, $\sqrt{p}=.05477$, $\log p=\bar{3}.47712$		$p=.01$, $\sqrt{p}=.1$, $\log p=\bar{2}.00000$		$p=.03$, $\sqrt{p}=.1732$, $\log p=\bar{2}.47712$		$p=.1$, $\sqrt{p}=.3162$, $\log p=\bar{1}.00000$	
Inches		c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$	c	$Q^{\frac{1}{3}}$
0.00	.0000	0.000	0	0.000	.0000	0.000	.0000	0.000	.0000	0.000	.0000	0.000	.0000	0.000	.0000	0.000	.0000	0.000	.0000	0.000	.0000	0.000	.0000
0.25	.5000	1.319	0	1.740	.0318	2.447	.0435	3.485	.0588	4.514	.0783	5.016	.0974	5.232	.1207	5.301	.1456	5.321	.1782	5.328	.2141	5.331	.2617
0.5	.7071	1.863	0	2.447	.0634	3.410	.0866	4.793	.1165	6.129	.1545	6.768	.1918	7.040	.2375	7.123	.2863	7.152	.3504	7.161	.4210	7.164	.5146
0.75	.8660	2.284	0	2.985	.0950	4.131	.1294	5.749	.1735	7.283	.2294	8.007	.2844	8.312	.3519	8.405	.4242	8.439	.5191	8.448	.6237	8.451	.7624
1.	1.000	2.637	0	3.435	.1265	4.726	.1720	6.525	.2300	8.204	.3034	8.987	.3756	9.316	.4646	9.416	.5599	9.452	.6852	9.462	.8232	9.466	1.006
1.5	1.225	3.230	0	4.182	.1894	5.700	.2567	7.769	.3417	9.646	.4489	10.51	.5548	10.87	.6858	10.98	.8263	11.02	1.011	11.03	1.215	11.04	1.485

FLOW-COEFFICIENTS *for* AIR-PASSAGES

BASED ON KUTTER'S FORMULA TRANSFORMED BY F. E. BRACKETT

$v = C\sqrt{RS} = \frac{C}{\sqrt{g}}\sqrt{RgS} = C_A\sqrt{R(gS)}$ ft. per sec.

$$C = \frac{41.6 + \frac{.00281}{gS} + \frac{1.811}{n}}{1 + \frac{\left(41.6 + \frac{.00281}{gS}\right)n}{\sqrt{R}}}$$ for English measure.

v = average velocity (ft. per sec.) throughout transverse section of passage.
C = Kutter's coefficient (transformed by Brackett).
R = transverse area (sq. ft.) divided by perimeter (ft.).
S = slope of fluid-gradient, figured in same manner as hydraulic-slope.
n = coefficient of surface roughness.

g = sp. gr. = 0.0012 for air at 70°F at sea-level, the basis for this table.
gS = loss of pressure due to flow per unit length of passage, measured by water column using same unit for height as for length.
To determine gS for other temperature or altitude for given weight of air moved, calculate as for 70°F and sea-level then increase gS thus found by 0.189% per added °F or by 4.7% per 1000 ft. of altitude. (Approximate rules.)
C_A = combined coefficient $\frac{C}{\sqrt{g}}$ = 28.8675 C for this table.
$gS = \frac{v^2}{C_A^2 R}$

Coefficient of roughness n	R	$\sqrt{R}$	$gS=0$ 0" per 100' $\sqrt{gS}=0$		$gS=.000025$ 1/32" per 100' $\sqrt{gS}=.005$		$gS=.00005$ 1/16" per 100' $\sqrt{gS}=.007071$		$gS=.000075$ 3/32" per 100' $\sqrt{gS}=.008660$		$gS=.0001$ 1/8" per 100' $\sqrt{gS}=.01$		$gS=.00015$ 3/16" per 100' $\sqrt{gS}=.01225$		$gS=.0002$ 1/4" per 100' $\sqrt{gS}=.01414$		$gS=.0003$ 3/8" per 100' $\sqrt{gS}=.01732$		$gS=.0004$ 1/2" per 100' $\sqrt{gS}=.02$		$gS=.0006$ 3/4" per 100' $\sqrt{gS}=.02449$		$gS=.0008$ 1" per 100' $\sqrt{gS}=.02828$	
			C_A	v	C_A	v	C_A	v	C_A	v	C_A	v	C_A	v	C_A	v	C_A	v	C_A	v	C_A	v	C_A	v
.012 quite smooth. Reasonably smooth sheathing or cement.	0.5	.7071	1701	0	2436	8.61	2699	13.5	2835	17.4	2918	20.6	3013	26.1	3067	30.7	3125	38.3	3156	44.6	3188	55.2	3205	64.1
	0.75	.8660	2083	0	2809	12.2	3049	18.7	3168	23.8	3240	28.1	3322	35.2	3367	41.2	3416	51.2	3442	59.6	3468	73.6	3482	85.3
	1.	1.	2406	0	3091	15.5	3303	23.4	3407	29.5	3468	34.7	3537	43.3	3575	50.6	3616	62.6	3638	72.8	3660	89.7	3671	104.
	1.5	1.225	2946	0	3508	21.5	3667	31.8	3741	39.7	3784	46.3	3833	57.5	3859	66.8	3887	82.5	3902	95.6	3917	118.		
	2.	1.414	3402	0	3816	27.0	3924	39.2	3973	48.7	4002	56.6	4033	69.9	4051	81.0	4068	99.6	4078	115.				
	2.5	1.581	3804	0	4059	32.1	4121	46.1	4149	56.8	4165	65.9	4183	81.0	4193	93.8	4202	115.						
	3	1.732	4167	0	4259	36.9	4280	52.4	4289	64.3	4293	74.4	4301	91.2	4304	105.								
	4	2.	4811	0	4575	45.7	4525	64.0	4503	78.0	4491	89.8	4477	110.										
	5	2.236	5379	0	4819	53.9	4709	74.5	4661	90.3	4635	104.												
	6.	2.449	5892	0	5017	61.4	4854	84.1	4785	102.														
	8	2.828	6804	0	5324	75.3	5074	101.																
	10	3.162	7607	0	5556	87.8	5236	117.																
.015 moderately smooth. Smooth masonry, e.g. ashlar or brick. Unsmoothed cement.	0.5	.7071	1361	0	1859	6.57	2052	10.3	2154	13.2	2218	15.7	2293	19.9	2335	23.4	2382	29.2	[illegible]	34.0	2433	42.1	2446	48.9
	0.75	.8660	1667	0	2162	9.36	2342	14.3	2434	18.3	2491	21.6	2556	27.1	2593	31.8	2633	39.5	2654	46.0	2676	56.8	2688	65.8
	1.	1.	1925	0	2396	12.0	2557	18.1	2638	22.8	2688	26.9	2744	33.6	2775	39.2	2809	48.7	2827	56.5	2846	69.7	2855	80.8
	1.5	1.225	2357	0	2748	16.8	2870	24.9	2930	31.1	2966	36.3	3006	45.1	3028	52.4	3052	64.7	3064	75.1	3077	92.3	3084	107.
	2.	1.414	2722	0	3012	21.3	3096	31.0	3137	38.4	3161	44.7	3187	55.2	3202	64.0	3217	78.8	3225	91.2	3234	112.		
	2.5	1.581	3043	0	3223	25.5	3272	36.6	3296	45.1	3309	52.3	3324	64.4	3332	74.5	3341	91.5	3346	106.				
	3	1.732	3333	0	3398	29.4	3416	41.8	3424	51.4	3428	59.4	3433	72.8	3436	84.2	3439	103.						
	4	2.	3849	0	3680	36.8	3639	51.5	3621	62.7	3610	72.2	3599	88.2	3592	102.								
	5	2.236	4303	0	3901	43.6	3809	60.2	3769	73.0	3746	83.8	3721	102.										
	6	2.449	4714	0	4082	50.0	3946	68.3	3886	82.4	3853	94.4	3817	115.										
	8	2.828	5443	0	4366	61.7	4154	83.1	4064	99.5	4014	114.												
	10	3.162	6086	0	4583	72.5	4309	96.4	4195	115.														

.020 slightly rough. Rough rubble. Dressed walls. Crude sheathing. Rough cement (gunnite).	0.5	.7071	1021	0	1318	4.66	1444	7.22	1513	9.27	1557	11.0	1609	13.9	1640	16.4	1673	20.8	1691	23.9	1711	29.6	1721	34.4
	0.75	.8660	1250	0	1549	6.71	1669	10.2	1733	13.0	1773	15.4	1820	19.3	1847	22.6	1877	28.3	1893	32.8	1909	40.3	1918	47.0
	1.	1.	1443	0	1730	8.65	1839	13.0	1897	16.4	1932	19.3	1974	24.2	1997	28.2	2023	35.0	2037	40.7	2051	50.2	2059	58.2
	1.5	1.225	1768	0	2009	12.3	2094	18.1	2137	22.7	2164	26.5	2194	32.9	2211	38.3	2230	47.3	2239	54.8	2250	67.3	2255	78.1
	2.	1.414	2041	0	2221	15.7	2282	22.8	2312	28.3	2330	32.9	2350	40.7	2362	47.2	2374	58.2	2381	67.3	2387	82.7	2391	95.6
	2.5	1.581	2282	0	2393	18.9	2431	27.2	2448	33.5	2459	38.9	2470	47.8	2477	55.4	2484	68.0	2488	78.7	2491	96.3	2493	111.
	3	1.732	2500	0	2541	22.0	2554	31.3	2560	38.4	2563	44.4	2567	54.3	2569	62.9	2572	77.2	2573	89.1	2574	109.		
	4	2.	2887	0	2779	27.8	2749	38.9	2734	47.4	2726	54.5	2717	66.6	2711	76.7	2706	93.7	2703	108.				
	5	2.236	3227	0	2969	33.2	2900	45.9	2868	55.5	2850	63.7	2829	77.5	2818	89.1	2806	109.						
	6	2.449	3536	0	3127	35.8	3023	52.4	2976	63.1	2948	72.2	2918	87.5	2902	101.								
	8	2.828	4082	0	3379	47.8	3214	64.3	3141	76.9	3099	87.7	3053	106.										
	10	3.162	4564	0	3576	56.5	3359	75.1	3264	89.4	3211	102.												
.025 moderately rough. Undressed but fairly even walls; usual with sedimentary rock.	0.5	.7071	816	0	1014	3.59	1102	5.51	1152	7.05	1184	8.37	1223	10.6	1246	12.8	1271	15.6	1285	18.2	1300	22.5	1308	26.2
	0.75	.8660	1000	0	1200	5.20	1285	7.87	1332	9.99	1362	11.8	1398	14.8	1419	17.4	1442	21.6	1454	25.2	1467	31.1	1474	36.1
	1.	1.	1155	0	1348	6.74	1427	10.1	1469	12.7	1496	15.0	1528	18.7	1546	21.9	1566	27.1	1577	31.5	1589	38.9	1595	45.1
	1.5	1.225	1414	0	1578	9.66	1640	14.2	1673	17.7	1694	20.7	1718	25.8	1731	30.0	1746	37.0	1754	43.0	1762	52.9	1767	61.2
	2.	1.414	1633	0	1756	12.4	1801	18.0	1824	22.3	1838	26.0	1855	32.1	1864	37.3	1874	45.9	1879	53.1	1885	65.3	1888	75.5
	2.5	1.581	1826	0	1903	15.0	1930	21.6	1944	26.6	1952	30.9	1962	38.0	1967	44.0	1973	54.0	1976	62.5	1979	76.6	1981	88.6
	3	1.732	2000	0	2028	17.6	2038	25.0	2043	30.6	2045	35.4	2049	43.5	2051	50.2	2053	61.6	2054	71.2	2055	87.2	2055	101.
	4	2.	2309	0	2235	22.3	2211	31.3	2200	38.1	2193	43.9	2185	53.5	2181	61.7	2176	75.4	2174	87.0	2171	106.		
	5	2.236	2582	0	2402	26.9	2348	37.1	2321	44.9	2306	51.6	2289	62.7	2279	72.1	2269	87.9	2264	101.				
	6	2.449	2828	0	2542	31.1	2459	42.6	2420	51.3	2398	58.7	2372	71.2	2358	81.7	2343	99.4	2336	114.				
	8	2.828	3266	0	2768	39.1	2636	52.7	2574	63.0	2539	71.8	2500	86.6	2479	99.2	2456	120.						
	10	3.162	3651	0	2948	46.6	2772	62.0	2691	73.7	2645	83.6	2595	101.	2568	115.								
.030 very rough. Surfaces somewhat jagged. Moderate exposed timbering.	0.5	.7071	680	0	821	2.90	887	4.43	924	5.66	949	6.71	979	8.48	996	9.76	1016	12.4	1027	14.5	1039	18.0	1045	20.9
	0.75	.8660	833	0	977	4.23	1041	6.37	1077	8.08	1100	9.53	1128	12.0	1144	14.0	1162	17.4	1172	20.3	1183	25.1	1188	29.1
	1.	1.	962	0	1101	5.51	1161	8.21	1194	10.3	1215	12.1	1240	15.2	1255	17.7	1271	22.0	1280	25.6	1289	31.6	1294	36.6
	1.5	1.225	1179	0	1297	7.94	1345	11.6	1371	14.5	1387	17.0	1406	21.1	1417	24.5	1429	30.3	1436	35.2	1443	43.3	1447	50.1
	2.	1.414	1361	0	1450	10.3	1485	14.9	1503	18.4	1515	21.4	1528	26.5	1536	30.7	1544	37.8	1549	43.8	1553	53.8	1556	62.2
	2.5	1.581	1521	0	1578	12.5	1599	17.9	1610	22.0	1617	25.6	1625	31.5	1629	36.4	1634	44.7	1636	51.7	1639	63.5	1641	73.4
	3	1.732	1667	0	1687	14.6	1695	20.8	1699	25.5	1701	29.5	1704	36.1	1705	41.8	1707	51.2	1708	59.2	1709	72.5	1709	83.7
	4	2.	1924	0	1870	18.7	1851	26.2	1841	31.9	1836	36.7	1829	44.8	1825	51.6	1821	63.1	1819	72.8	1817	89.0	1816	103.
	5	2.236	2152	0	2018	22.6	1975	31.2	1953	37.8	1940	43.4	1926	52.7	1917	60.6	1909	73.9	1904	85.1	1899	104.		
	6	2.449	2357	0	2144	26.3	2077	36.0	2045	43.4	2026	49.6	2004	60.1	1992	69.0	1979	84.0	1972	96.6	1965	118.		
	8	2.828	2722	0	2350	33.2	2241	44.8	2189	53.6	2159	61.1	2125	73.6	2106	84.2	2086	102.	2076	117.				
	10	3.162	3043	0	2515	39.8	2368	53.0	2300	63.0	2260	71.5	2216	85.8	2192	98.0	2166	119.						
.040 extremely rough. Surfaces very jagged, such as with igneous rock. Heavy exposed timbering.	0.5	.7071	510	0	592	2.09	632	3.16	656	4.02	671	4.74	691	5.98	702	7.02	715	8.76	723	10.2	731	12.7	735	14.7
	0.75	.8660	625	0	709	3.07	749	4.59	772	5.79	787	6.82	805	8.54	816	9.99	828	12.4	835	14.5	842	17.9	846	20.7
	1.	1.	722	0	803	4.02	841	5.95	862	7.47	876	8.76	893	10.9	903	12.8	914	15.8	920	18.4	927	22.7	930	26.3
	1.5	1.225	884	0	954	5.84	985	8.53	1002	10.6	1013	12.4	1026	15.4	1034	17.9	1043	22.1	1047	25.6	1052	31.6	1055	36.5
	2.	1.414	1021	0	1074	7.59	1097	11.0	1109	13.6	1117	15.8	1126	19.5	1132	22.6	1138	27.9	1141	32.3	1145	39.7	1146	45.8
	2.5	1.581	1141	0	1175	9.29	1189	13.3	1196	16.4	1201	19.0	1207	23.4	1210	27.1	1214	33.2	1215	38.4	1218	47.2	1219	54.5
	3	1.732	1250	0	1263	10.9	1267	15.5	1270	19.1	1272	22.0	1274	27.0	1275	31.2	1276	38.3	1277	44.2	1278	54.2	1278	62.6
	4	2.	1443	0	1410	14.1	1397	19.8	1391	24.1	1386	27.7	1382	33.9	1379	39.0	1376	47.7	1374	55.0	1373	67.3	1372	77.6
	5	2.236	1614	0	1532	17.1	1502	23.7	1487	28.8	1477	33.0	1466	40.1	1460	46.2	1453	56.3	1450	64.8	1446	79.2	1444	91.3
	6	2.449	1768	0	1637	20.0	1590	27.5	1567	33.2	1552	38.0	1536	46.1	1526	52.9	1516	64.3	1511	74.0	1505	90.3	1502	104.
	8	2.828	2041	0	1810	25.6	1733	34.7	1695	41.5	1671	47.3	1645	57.0	1630	65.2	1615	79.1	1606	90.8	1597	111.		
	10	3.162	2282	0	1951	30.8	1846	41.3	1795	49.2	1764	55.8	1729	67.0	1710	76.5	1689	92.5	1678	106.				

Data pertaining to interpolation:— C_A and $v = 0$ when $R = 0$. For low value of R and constant gS, C_A varies approximately as $\sqrt{R}$. For any value of gS and constant R, v varies approximately as $\sqrt{gS}$. C_A changes but slightly for increase in gS above .0008.

Compiled by C. S. Witherell, July 1926

$Q = 69.44 \div 0.03130 = 2219$; $Q^{1/3} = 13.04$, and
$p_L = (0.5 \div 16) \div 100 = 0.0003125$.
d found by interpolation as in Ex. C. 1. $= 44.79$
c found by interpolation as in Ex. C. 1. $= 27.50$
$\text{Log } d = 0.4(\log 2219 - \log 27.50) - 0.2 \log 0.0003125 + 0.18723 = 1.65099$

$d = 44.77$, a close check.

By using the equation:

$\text{Log } p_L = 2(\log Q - \log c) - 5 \log d + 0.93617$ and assuming $c = 27.50$, then:

If $d = 44$ be chosen, $p_L = 0.0003409$;
If $d = 46$ be chosen, $p_L = 0.0002729$.

If this example be solved for sea level it will be found that $d = 42.09$. This suggests the approximate rule for handling a gas compressed to not over say 3 lb. gage pressure, d should be increased 0.64 per cent. per 1000 ft. of altitude above sea level under the conditions that W, p_L, n and gage pressure remain unchanged.

Example C. 2. b.—Same as Example C. 2. a, except blast is preheated to 570° F.

$$g = 0.00098 \times \frac{459 + 70}{459 + 570} = 0.0005038; \sqrt{g} = 0.02245$$

$W = 69.44$ lb. per sec.
$Q = 69.44 \div 0.02245 = 3093$; $Q^{1/3} = 14.57$.
$p_L = 0.0003125$
d found by interpolation as in Example C. 1. $= 50.73$
c found by interpolation as in Example C. 1. $= 28.11$
$\text{Log } d = 0.4 (\log 3093 - \log 28.11) - 0.2 \log 0.0003125 + 0.18723 = 1.70487$

$d = 50.68$, a close check.

Formula for Handling a Heated Gas

This example suggests the following approximate formula for handling a heated gas under the conditions that W, p_L, n and gage pressure remain unchanged:

$$\begin{aligned} d_H &= d_1[1 + 0.00221(t^\circ \text{ F.} - 70)]^{1/6} \\ &= d_1[1 + 0.0040\ (t^\circ \text{ C.} - 21)]^{1/6} \\ &= d_1K \end{aligned}$$

wherein,

d_H = diameter for heated gas;
d_1 = diameter for 70° F. or 21° C.;
t° F. = temperature, Fahrenheit, of heated gas;
t° C. = temperature, Centigrade, of heated gas;
K = thermal factor.

Upon this formula is based the following table:

$t°$ C.	$t°$ F.	K.
100	212	1.046
200	392	1.094
300	572	1.133
500	932	1.195
700	1292	1.244
1000	1832	1.304
1500	2732	1.380

Conduits intended for conveying very hot gases are usually lined with refractory material, which is somewhat rougher than material having $n = 0.012$. For worn-brick lining n will probably be as high as 0.015 in which case d should be again increased about 10 per cent. if it is desired to hold p_L nearly at its given value.

Calculation of Diameter of a Converter Blast Main

Example C. 3.—Required the diameter of a converter blast main to deliver 15,000 cu. ft. of standard free air per min. at a gage pressure of 14.7 lb. per sq. in. and an allowable friction loss of 3 lb. per 1000 ft.

$$g = \frac{14.7 + 14.7}{14.7} \times 0.0012 = 0.0024, \text{ and } \sqrt{g} = 0.04899$$

$W = (15000 \times 62.4 \times 0.0012) \div 60 = 18.72$ lb. per sec.
$Q = 18.72 \div 0.04899 = 382.1;\ Q^{1/6} = 7.256$
$p_L = 0.003$
d found by interpolation as in Example C. 1. $= 15.31$
c found by interpolation as in Example C. 1. $= 22.34$
$\text{Log } d = 0.4 (\log 382.1 - \log 22.34) - 0.2 \log 0.003 + 0.18723 = 1.18505$
$d = 15.31$, an exact check.
Assuming $c = 22.34$, then, if $d = 16$ be chosen, $p_L = 0.002409$.

Saving in Friction Loss by Using Larger Pipe

Example C. 4.—Required the diameter of a compressed-air line to handle 1000 cu. ft. standard free air per min. at a gage pressure of 100 lb. per sq. in. and an allowable friction-loss of 1.5 lb. per 100 ft.

$$g = \frac{100 + 14.7}{14.7} \times 0.0012 = 0.009363, \text{ and } \sqrt{g} = 0.09676$$

$W = (1000 \times 62.4 \times 0.0012) \div 60 = 1.248$ lb. per sec.
$Q = 1.248 \div 0.09676 = 12.90;\ Q^{1/6} = 2.345$
$p_L = 0.015$.
d found by interpolation as in Example C. 1. $= 3.40$
c found by interpolation as in Example C. 1. $= 14.66$
$\text{Log } d = 0.4 (\text{Log } 12.90 - \log 14.66) - 0.2 \log 0.015 + 0.18723 = 0.52980$
$d = 3.387$, a close check.
Assuming $d = 3.068$ (diameter of standard 3-in. pipe), then by interpolation between $Q^{1/6}$ values $\log p_L = \bar{2}.41540$; $p_L = 0.02603$.

Corresponding, c by interpolation = 14.17.

Log p_L = 2 (log 12.90 − log 14.17) − 5 log 3.068 + 0.93617 = $\bar{2}.42031$

p_L = 0.02632, a fair check.

Assuming d = 4.026 (diameter of standard 4-in. pipe), then by interpolation between $Q^{1/8}$ values log p_L = $\bar{3}.74135$; p_L = 0.005513.

Corresponding, c by interpolation = 15.45.

Log p_L = 2 (log 12.90 − log 15.45) − 5 log 4.026 + 0.93617 = $\bar{3}.75514$.

p_L = 0.005690, a rough check.

The foregoing illustrates the enormous saving in friction loss sometimes obtainable by choosing the next larger size of commonly used pipe.

Rate of Flow in Pipe Connecting Two Compressed Air Receivers

Example C. 5.—A standard ¾-in. pipe without sharp bends or other obstructions is used for connecting two compressed air receivers. The gage pressure in the first receiver is 100 lb. per sq. in., in the second 30 lb. per sq. in. Required the rate of flow, ignoring change of velocity head and other orifice effects.

As a gas flows in a pipe, it must lose pressure due to friction which causes a decrease in specific gravity, which in turn means an increase in volume and velocity. The weight per sec. flowing past each point of the pipe is however the same throughout.

Since the specific gravity of the flowing gas changes, p_L must vary from beginning to end of pipe. Where the drop in pressure is relatively slight, this effect may be safely ignored as was done in the preceding examples.

Let P = absolute pressure at any point;
P_1 = absolute pressure at beginning;
P_2 = absolute pressure at end;
L = length of pipe;
g = sp. g. at any point;
g_0 = sp. g. of standard free gas;

then:

$$p_L = \frac{\partial P}{\partial L}, \text{ and } g = g_0\left(\frac{P}{14.7}\right)$$

$$W = 0.3403\, c\sqrt{d^5\, g p_L} = 0.08876\, c\sqrt{d^5\, g_0 P\frac{\partial P}{\partial L}}$$

$$P\partial P = \left[\frac{W^2}{(0.08876c)^2\, d^5 g_0}\right]\partial L$$

$$\int_{P_2}^{P_1} P\partial P = \int_0^L\left[\frac{W^3}{(0.08876c)^2\, d^5 g_0}\right]\partial L$$

$$\frac{P_1^2 - P_2^2}{2} = \left[\frac{W^2}{(0.08876c)^2 d^5 g_0}\right]L$$

$$W = 0.06276c\sqrt{d^5 g_0}\sqrt{\frac{P_1^2 - P_2^2}{L}}.$$

This last equation may be written:

$$W = 0.3403c\sqrt{d^5\left(\frac{P_1+P_2}{2\times 14.7}g_0\right)\left(\frac{(P_1-P_2)}{L}\right)}$$

wherein it is evident that:

$\frac{P_1+P_2}{2\times 14.7}g_0$ = mean value of g.

$\frac{P_1-P_2}{L}$ = average value of p_L throughout length of pipe.

With this understanding of the values to be given g and p_L, the equation assumes its original form, namely, $W = \sqrt{g}(0.3403c\sqrt{d^5p_L})$.

In this example

$$\begin{aligned} d &= 0.824 \text{ (diameter of standard } \tfrac{3}{4}\text{-in. pipe)} \\ P_1 &= 100 + 14.7 = 114.7 \\ P_2 &= 30 + 14.7 = 44.7 \\ L &= 500 \\ g_0 &= 0.0012 \text{ (\textit{i. e.} at 14.7 lb. pressure and 70° F.)} \end{aligned}$$

c will practically not vary with p_L for p_L is greater than 0.1 throughout. Interpolation for c given, $d = 0.824$.

d	$\sqrt{d}$	c
0.750	0.8660	8.451
0.824	0.9077	x = 8.767
1.000	1.0000	9.466

$$0.06276c\sqrt{d^5 g_0} = 0.01175$$

$$\sqrt{\frac{P_1^2 - P_2^2}{L}} = 4.724$$

$$W = 0.01175 \times 4.724 = 0.05551 \text{ lb. per sec.}$$

$$V_0 = \frac{0.05551}{62.4 \times 0.0012} = 0.7413 \text{ cu. ft. standard free air per sec.}$$

$$= 44.48 \text{ cu. ft. standard free air per min.}$$

Calculation of Diameter of Pipe Handling Free Hydrogen

Example C. 6.—Required the diameter of a pipe handling 25 cu. ft. of free hydrogen per min. at 6 in. water-pressure and at an allowable friction loss of 1 in. per 100 ft.

The sp. g. of a perfect gas,

$$g = 0.00009 \times \frac{273}{T} \times \frac{B}{760} \times \frac{M}{2} = 0.00001616\,\frac{BM}{T},$$

wherein:

T = absolute temperature expressed in °C;
B = barometric pressure expressed in mm. of mercury column;
M = molecular weight of the gas.

Converted into English measure the equation becomes:

$$g = \frac{0.001504PM}{459 + t°\text{F.}},$$

wherein:

P = absolute pressure expressed in lb. per sq. in.;

$t°$ F. = temperature Fahrenheit. The absolute zero is at $-459°$ F.

The molecular weight of most (but not all) elemental gases is twice the atomic weight.

As an example for a mixed gas the following calculation for air is given:

$$\begin{aligned} &20.8 \text{ per cent. by volume of } O_2 \text{ at } 32 = 6.656 \\ &79.2 \text{ per cent. by volume of } N_2 \text{ at } 28 = \underline{22.176} \\ &\qquad\qquad\qquad\qquad\qquad\qquad M = 28.8 \end{aligned}$$

Hence for standard free air at 70° F.:

$$g = \frac{0.001504 \times 14.7 \times 28.8}{459 + 70} = 0.001204$$

The rounded-off value 0.0012 was used in the preceding examples. For this example:

$$P = 14.7 + \frac{6}{12 \times 2.308} = 14.92$$

$M = 2$, and $t°$ F. = 70

$$g_0 = \frac{0.001504 \times 14.7 \times 2}{529} = 0.00008359$$

$$g = \frac{0.001504 \times 14.92 \times 2}{529} = 0.00008484$$

$\sqrt{g} = 0.009211$

$W = (25 \div 60) \times 0.00008359 \times 62.4 = 0.002173$ lb. per sec.

$Q = 0.002173 \div 0.009211 = 0.2359$; $Q^{2/5} = 0.6179$

$$p_L = \frac{1}{12 \times 2.308 \times 100} = 0.000361.$$

d found by interpolation as in Example C. 1 = 1.628

c found by interpolation, using $\sqrt{d}$ and

log p_L values = 10.89.

Log d = 0.4 (log 0.2359 − log 10.89) − 0.2 log 0.000361 + 0.18723 = 0.21001

d = 1.622, a fair check.

Assuming d = 1.610 (diameter of standard 1½-in. pipe) and c = 10.89, then:

Log p_L = 2(log 0.2359 − log 10.89) − 5 log 1.610 + 0.93617 = $\bar{4}.57342$

p_L = 0.0003745, equivalent to 1.04 in. water-pressure per 100 ft.

In general, for all non-viscous fluids d is approximately inversely proportional to $\sqrt[6]{g}$ when W, p_L and n remain constant; but only roughly so for very widely divergent values of g and high values of p_L.

Calculation of Diameter of a Steam Main

Example C. 7.—Required the diameter of a steam-main handling 54,000 lb. of saturated steam per hr. at 164.7 lb. per sq. in. absolute. Allowable friction loss 1 lb. per sq. in. per 100 ft.

One cu. ft. of saturated steam at 164.7 lb. per sq. in. absolute (*i. e.*, 150 lb. gage pressure at sea level) weighs 0.3627 lb., hence:

$g = 0.3627 \div 62.4 = 0.005812$, and $\sqrt{g} = 0.07624$

$W = 54000 \div (60 \times 60) = 15.00$ lb. per sec.

$Q = 15.00 \div 0.07624 = 196.7$; $Q^{1/3} = 5.816$

$p_L = 0.01$.

d found by interpolation as in Example *C*. 1. $= 9.658$

c found by interpolation as in Example *C*. 1 $= 19.94$

$\text{Log } d = 0.4 (\log 196.7 - \log 19.94) - 0.2 \log 0.01 + 0.18723 = 0.98486$

$d = 9.657$, a close check.

A good quality of steam-pipe would have a surface coefficient $n = 0.010$, which would permit reducing d about 8 per cent. without changing the other given conditions. Hence a standard 9-in. pipe would suffice.

Calculation of Diameter of a Steam Exhaust Main

Example C. 8.—Required the diameter of a steam exhaust main to a central condenser, to handle 54,000 lb. of exhaust steam at a mean absolute pressure of 1.433 lb. per sq. in. (27-in. vacuum). Allowable average friction loss, 0.25 lb. (½-in. vacuum) per 100 ft.

One cubic foot of saturated steam at 1.433 lb. per sq. in. absolute weighs 0.00420 lb., hence:

$g = 0.00420 \div 62.4 = 0.00006731$, and $\sqrt{g} = 0.008204$

$W = 54,000 \div (60 \times 60) = 15.00$ lb. per sec.

$Q = 15.00 \div 0.008204 = 1828$; $Q^{1/3} = 12.23$

$p_L = 0.0025$.

d found by interpolation as in Example *C*. 1. $= 28.27$

c found by interpolation as in Example *C*. 1. $= 25.52$

$\text{Log } d = 0.4 (\log 1828 - \log 25.52) - 0.2 \log 0.0025 + 0.18723 = 1.44968$

$d = 28.16$, a fair check.

Considering the better n value that would probably obtain, a 28-in. pipe would suffice.

This and the preceding example show somewhat the relationship between high-pressure and condenser mains handling the same amount of steam.

Calculation of Diameter of Pipe for Handling Heavy Leach Solution

Example C. 9.—Required the diameter of a pipe for handling 4000 gal. per min. of heavy leach solution. $g = 1.25$, $p_L = 0.002$.

$W = (4000 \times 8.345 \times 1.25) \div 60 = 695.4$ lb. per sec.
$\sqrt{g} = 1.118$; $Q = 695.4 \div 1.118 = 622.0$; $Q^{1/5} = 8.536$
d found by interpolation as in Example C. 1. $= 19.83$
c found by interpolation as in Example C. 1. $= 23.66$
$\text{Log } d = 0.4 \ (\log\ 622.0 - \log\ 23.66) - 0.2 \ \log\ 0.002 + 0.18723 = 1.29494$
$d = 19.72$, a fair check.

This example illustrates an actual case encountered in a large copper leaching plant. The principal solutions were handled by centrifugal pumps fitted with direct connected a. c. constant-speed motors. The solution was delivered through 16-in. pipes to elevated tanks, the most remote of which was about 1000 ft. away.

As the pumps did not deliver their rated capacity to the remote tanks and tests indicated that there was not sufficient surplus head above static and velocity heads to overcome friction, in other words 16-in. pipe was too small, 24-in. pipe was installed.

In order to show the benefits to be expected by substituting the 24- for 16-in. pipe the following calculations are made, the pump characteristics being assumed:

	Feet
Total head produced by pump (assumed to be constant)	52.0
Static head	45.0
Available for velocity and friction heads	7.0

By previous assumption and correction it was found that 16-in. pipe figures to 69.1 per cent. of rated capacity.

$$V = (0.691 \times 4000 \times 0.1337) \div 60 = 6.16 \text{ cu. ft. per sec.}$$

$$v = 6.16 \div \frac{\pi \overline{16}^2}{4 \times 144} = 4.41 \text{ ft. per sec.}$$

$$\text{Velocity head} = \frac{v^2}{64.4} = 0.302 \text{ ft.}$$

Friction head, other than that due to pipe friction = say 2.5 ft. at full capacity or $2.5 \times \overline{0.691}^2 = 1.194$ ft. at 69.1 per cent. capacity.

Head available for pipe friction:

$$7.000 - 0.302 - 1.194 = 5.504 \text{ ft.}$$

Pressure available for pipe friction:

$$(5.504 \times 62.4 \times 1.25) \div 144 = 2.981 \text{ lb. per sq. in.}$$

Assuming 1000 ft. of pipe then $p_L = 0.002981$.
By interpolation $c = 22.58$
$\text{Log } 16 = 0.4(\log Q - \log 22.58) - 0.2 \log 0.002981 + 0.18723$
$\text{Log } Q = 2.63313$; $Q = 429.7$
$W = Q\sqrt{g} = 480.4$ lb. per sec.

$$\text{Volume} = \frac{480.4 \times 60}{8.345 \times 1.25} = 2763 \text{ gal. per min.,}$$

which is the flow to be expected through 1000 ft. of 16-in. pipe.

By previous assumption and correction it was found that 24-in. pipe figures to 134.0 per cent. rated capacity.

$V = (1.34 \times 4000 \times 0.1337) \div 60 = 11.94$ cu. ft. per sec.

$$v = 11.94 = \frac{\pi\overline{42}^2}{4 \times 144} = 3.80 \text{ ft. per sec.}$$

$$\text{Velocity head} = \frac{v^2}{64.4} = 0.224 \text{ ft.}$$

Friction head other than that due to pipe friction $= 2.5 \times \overline{1.34}^2 =$ 4.489 ft.

Head available for pipe friction:

$$7.000 - 0.224 - 4.489 = 2.287 \text{ ft.}$$

Pressure available for pipe friction:

$$(2.287 \times 62.4 \times 1.25) \div 144 = 1.239 \text{ lb. per sq. in.}$$

Assuming 1000 ft. of pipe, then $p_L = 0.001239$

By interpolation $c = 24.62$.

Log $24 = 0.4(\log Q - \log 24.62) - 0.2 \log 0.001239 + 0.18723$.

Log $Q = 2.92027$; $Q = 832.3$.

$W = Q\sqrt{g} = 930.5$ lb. per sec.

$$\text{Volume} = \frac{930.5 \times 60}{8.345 \times 1.25} = 5352 \text{ gal. per min.,}$$

which is the flow to be expected through 1000 ft. of 24-in. pipe.

Motor power, drop in total produced head and other limitations would probably not permit obtaining this high capacity, but the rated capacity of 4000 gal. per min. would doubtless be obtainable.

Examples of Air-pressure Calculations Illustrating Use of Flow Coefficients for Air Passages Table

Example A. 1.—Required the loss in pressure due to the flow of 750 cu. ft. of air (at 70° F. and sea level) per sec. in a rectangular underground passage 8 by 10 ft. rough rock walls ($n = 0.035$).

$$R = 80 \div 36 = 2.22$$
$$v = 750 \div 80 = 9.375 \text{ ft. per sec.}$$

Since the values $v = 9.375$, $R = 2.22$ and $n = 0.035$ can not be taken directly from the table Flow Coefficients for Air Passages, it is necessary to interpolate between the values $v = 0.0$ and $v = 10.3$, which appear in the line for $R = 2$ in the section for $n = 0.030$. This gives the value $C_A = 1442$. Similarly, interpolating in the $R = 2.5$ gives $C_A = 1564$. Next, interpolating between these two lines for $R = 2.22$ gives the tabular value for C_A of 1496 for $n = 0.03$. Performing a similar set of operations for these values in the section of the table for $n = 0.04$ gives a value of

C_A of 1125. Interpolating between these two values of C_A gives a final value of $C_A = 1335$ for $n = 0.035$, $R = 2.22$ and $v = 9.375$.

$$gS = \frac{v^2}{C_A{}^2R} = \left(\frac{9.375}{1335}\right)^2 \div 2.22 = 0.0000222$$

$$= \text{about } {}^{17}\!/_{64}\text{-in. water pressure per 1000 ft.}$$

Saving in Friction Obtained by Sheathing a Rectangular Passage

Example A. 2.—What is the saving in friction to be obtained by sheathing a rectangular passage handling 50,000 cu. ft. of air (at 70° F. and sea level) per min.?

Unsheathed passage: 7 by 6 ft., $n = 0.030$.
Sheathed passage: 6 ft. 6 in. by 5 ft. 6 in., $n = 0.012$.
Unsheathed passage:

$$R = \frac{42}{26} = 1.615;$$

$$v = \frac{50000}{60 \times 42} = 19.84 \text{ ft. per sec.}$$

C_A found by interpolation as in Example *A.* 1. = 1425.

$$gS = \left(\frac{19.84}{1425}\right)^2 \div 1.615 = 0.0001200$$

Sheathed passage:

$$R = \frac{6.5 \times 5.5}{24} = 1.490;$$

$$v = \frac{50000}{60 \times 6.5 \times 5.5} = 23.31 \text{ ft. per sec.}$$

C_A found by interpolation as in Example *A.* 1. = 3536

$gS = \left(\frac{23.31}{3536}\right)^2 \div 1.490 = 0.0000292$ (versus 0.0001200 for unsheathed passage).

This illustrates the great advantage of sheathing (or otherwise making smooth) a ventilating duct, even if by so doing the transverse area be somewhat reduced.

Fan Pressure in Mine with Two Sets of Workings

Example A. 3.—Assume a mine made up as follows: Main passages to and from workings 1000 ft. of 8 by 14 ft., roughness $n = 0.020$. Two sets of workings taking air in parallel. First set composed of 400 ft. of 7 by 7 ft.-passage, 500 ft. of 12 by 25-ft. passage, roughness $n = 0.030$. Air requirement 25,000 cu. ft. standard per minute. Second set composed of 200 ft. of 7 by 7-ft. passage, 600 ft. of 12 by 40-ft. passage, roughness $n = 0.035$. Air requirement 45,000 cu. ft. standard per min. Allow 50 per cent. additional pressure for bends and miscellaneous obstacles. What fan pressure is required?

Main passages:

$$R = \frac{112}{44} = 2.55;$$

$$v = \frac{70000}{60 \times 112} = 10.42 \text{ ft. per sec.}$$

$$C_A \text{ (by interpolation)} = 2361;$$

$$gS = \left(\frac{10.42}{2361}\right)^2 \div 2.55 = 0.00000764.$$

Pressure for 1000 ft. of passage = 0.00000764 × 1000 × 12 = 0.0917-in. water-column.

First branch, 7 by 7-ft. passage:

$$R = \frac{49}{28} = 1.75;$$

$$v = \frac{25000}{60 \times 49} = 8.503 \text{ ft. per sec.}$$

$$C_A \text{ (by interpolation)} = 1369;$$

$$gS = \left(\frac{8.503}{1369}\right)^2 \div 1.75 = 0.00002204.$$

Pressure for 400 ft. of passage = 0.00002204 × 400 × 12 = 0.1058-in. water-column.

First branch, 12 by 25-ft. passage:

$$R = \frac{300}{74} = 4.05;$$

$$v = \frac{25000}{60 \times 300} = 1.389 \text{ ft. per sec.}$$

$$C_A \text{ (by interpolation)} = 1930;$$

$$gS = \left(\frac{1.389}{1930}\right)^2 \div 4.05 = 0.000000128.$$

Pressure for 500 ft. of passage = 0.000000128 × 500 × 12 = 0.0008-in. water column.

Pressure for first branch = 0.1058 + 0.0008 = 0.1066-in. water-column.

Second branch, 7 by 7-ft. passage:

$$R = \frac{49}{28} = 1.75;$$

$$v = \frac{45000}{60 \times 49} = 15.31 \text{ ft. per sec.}$$

$$C_A \text{ (by interpolation)} = 1251;$$

$$gS = \left(\frac{15.31}{1251}\right)^2 \div 1.75 = 0.00008561.$$

Pressure for 200 ft. of passage = 0.00008561 × 200 × 12 = 0.2055-in. water-column.

Second branch, 12 by 40-ft. passage:

$$R = \frac{480}{104} = 4.615;$$

$$v = \frac{45000}{60 \times 480} = 1.563$$

$$C_A \text{ (by interpolation)} = 1490;$$

$$gS = \left(\frac{1.563}{1490}\right)^2 \div 4.615 = 0.000000238.$$

Pressure for 600 ft. of passage, $0.000000238 \times 600 \times 12 = 0.0017$-in. water-column.

Pressure for second branch $= 0.2055 + 0.0017 = 0.2072$-in. water-column.

Where branches are in parallel the highest branch pressure determines the total fan-pressure. In this case the second branch governs and therefore the first branch must be throttled to divide the flow of air as specified.

Summary:

	Water Column Inches
Main passages	0.0917
Second branch	0.2072
	0.2989
Fifty per cent. for bends, etc.	0.1495
Total fan pressure	0.4484
or about	7/16

In order to deliver the required weight of air for other than standard conditions, increase this pressure by 4.7 per cent. per 1000 ft. of altitude above sea level, and if temperature is greater or less than 70° F., increase or decrease pressure by 0.189 per cent. per degree F. (these rules are only approximate).

W. S. Weeks, Berkeley, Cal. (written discussion).—The author does not propose to modify the fundamental formula for loss of pressure in ducts, but proposes to use it in another form.

The fundamental formula for loss of head in a duct where the flow is turbulent is $h = \frac{flv^2}{R2g}$.

h = loss of head in feet.
l = length of duct in feet.
v = velocity in ft. per sec.
R = hydraulic radius.
g = acceleration due to gravity.
f = friction factor.

For ventilation work this formula is transformed into $P = \frac{KSV^2}{A}$.

P = loss of pressure, lb. per sq. ft.
S = rubbing surface, sq. ft.
V = velocity, ft. per min.
A = cross-sectional area of duct, sq. ft.
K = friction factor.

In the transformation from head in feet to pressure in pounds per square foot, a factor w, which is the density of the air in pounds per cubic foot, is introduced. This w is combined with the true friction factor. All the constants necessary in the conversion are also in the factor K.

If w is factored out of K, and the new factor with w left out is denoted by n, the formula becomes $P = \frac{nwSV^2}{A}$.

This formula may then be thrown into the form of the Chezy or slope formula, where p = perimeter of duct: $\frac{P}{w} = \frac{nSV^2}{A} = \frac{nplV^2}{A}$. Since $\frac{P}{w} = h$ = head in feet, then $h = \frac{nplV^2}{A}$, and $V^2 = \frac{Ah}{npl}$. Since $\frac{h}{l}$ = slope = S in the slope formula, and $\frac{A}{p}$ = hydraulic radius = R, by substituting these values $V^2 = \frac{RS}{n}$. If v is feet per second, then $3600v^2 = \frac{RS}{n}$, and $v = \frac{1}{60}\sqrt{\frac{1}{n}}\sqrt{RS}$.

This is the slope formula and $C = \frac{1}{60}\sqrt{\frac{1}{n}}$ or $\frac{1}{60}\sqrt{\frac{w}{K}}$ which is the form given by the author. The relation between f and n is

$$n = \frac{f}{2g \times 3600}$$

Kutter's formula is an empirical one for determining the value of C. The reason for determining C in ventilation work is to make it possible to predict the drop in pressure in a duct with a given flow. The formula contains as one of its parts the pressure drop that it is desired to find. A solution can of course be obtained by approximations. The question arises: Should we use an empirical formula of this sort when it may be possible to develop a method for determining the value of the friction factor in the ordinary ventilation formula by using certain principles that have been found to be true?

The work of Reynolds, Blausius, and Stanton and Pannell demonstrated that for ducts, the cross-sections of which are geometrically similar, the factor f is a function of the expression $\frac{lv}{z}$ where l is any dimension of the duct, v is the velocity and z is the kinematic viscosity (viscosity divided by density). If f is plotted against $\frac{lv}{z}$ the friction factor for any liquid or gas may be determined by calculating the value of $\frac{lv}{z}$. In ventilation work it would be necessary to plot such a curve for each geometrical shape and for surfaces of different kinds.

The doctrine of similarity extends even to the irregularities of the surface that constitute roughness, that is if a large duct is to be similar to

a small duct its roughness should be exaggerated as well as its dimensions. A large mine opening is no rougher than a small one so it would not be possible to predict exactly from an experiment on a small drift what the friction factor for the large drift would be. It is possible that other factors might be introduced to take care of the discrepancy or it is possible that sufficiently accurate results may be obtained if it is ignored.

The method outlined will undoubtedly come into general use as the method for handling the problem of fluid friction in ducts. It would seem reasonable that friction in ventilation work should be handled in the same manner. Curves developed would be applicable to any liquid or gas if the value of the kinematic viscosity is known. Many curves for circular ducts are already in existence.

O. Singstad,* New York, N. Y. (written discussion).—The opinion seems to prevail among most ventilating engineers that air is so different from water that none of the laws governing the latter can be applied to the former. The study given to our test data has convinced me that, after due consideration is given to the difference in densities of the two fluids, the two act exactly alike and obey the same laws. If the author succeeds in proving that the Kutter and Chezy formulas can be applied to air flow he will have gone a long way in dispelling the present uncertainty as to the laws of air flow and establishing the same degree of confidence which exists in computing the flow of water.

The revised Kutter formula was applied to a number of the tests on the concrete model duct and the computed value of C compared with the actual value obtained from the tests. The agreement was not as close as might be desired using a value of $N = 0.010$ as suggested by the author. I believe $N = 0.011$ to be more representative of the actual duct surface and this value brings the constant C well within the permissible error. However, a value of $N \doteq 0.0115$ gives practically exact agreement throughout the full range of velocities from 35 f.p.s. to 95 f.p.s.

I believe that further study of original test data, rather than of empirical formulas derived therefrom, might show some very interesting results.

G. E. McElroy, Butte, Mont. (written discussion).—Although this paper is interesting it is not practical for the mine ventilation engineer because the degree of accuracy of application to mine ventilation conditions does not warrant such elaborate calculations. The Atkinson formula is no more complex than the slope formula and is just as easily applied.

Application of the Kutter formula for computing the constant C of the slope formula is much more laborious than estimating the constant

* Chief engineer, The Hudson Tunnel.

K of the Atkinson formula and, for the average mine ventilation problem it is no more accurate. In practically every problem roughness of surfaces is the controlling factor and its effect must be estimated in either case.

The effect of density is as easily handled by one formula as by the other but it is more likely to be disregarded in the case of the Atkinson formula.

"H = pressure in feet of fluid flowing" should be added to the group of notations on the first page.

The data regarding critical radius is interesting but a value of 3.28 or over for R would seldom, if ever, be met in mine work.

In the discussion of the variations in coefficients, or friction factors, the predominant effect of roughness in reducing the effect of velocity changes should have been mentioned.

The general theory of fluid flow would indicate that the transformation of Kutter's formula from water to gases would involve the kinematic viscosity, but, as values of the latter change relatively slowly with change of temperature of both air and water, perhaps no great error is involved. The water-air ratio, however, is reduced to but 15.5 for conditions that give a density ratio of 834. The ratio of change in flow conditions is, therefore, but a small fraction of that indicated by the author.

A. S. Richardson, Butte, Mont. (written discussion).—In my work, I seldom use friction coefficients or calculations based on them. It has been my practice, as far as possible, to traverse the mine workings with a portable gage, and measure the pressure losses as they occur. In some cases, as where a new shaft is to be opened, it may be necessary to make calculations based on friction factors; but even in such problems, it is necessary to use considerable judgment, because even though the exact friction factor for the proposed opening should be known the physical condition of an airway does not remain constant, and satisfactory ventilation can be expected only by modifying the results of calculations on the basis of experience, as to what is likely to occur.

Economic Design of Mine Airways

By A. S. Richardson,* Butte, Mont.

(New York Meeting, February, 1926)

The design of mine airways receives, in general, very little engineering treatment. To a large extent this is, of course, due to the fact that information upon which to base calculations is seldom available in definite form. In many mines the extent of the workable deposits may not be known at the time an air course is planned, and the length of service is indeterminable. Volume of air required for ventilation may also increase with extension of the life of a property, and natural obstacles, such as heavy ground, often place limitations upon the size of opening that can be maintained.

Perhaps most of the shafts used in the ventilation of metal mines were originally intended as hoisting shafts, and many of the tunnels and crosscuts similarly used, were also designed as haulage ways or for other operating purposes. Even when a shaft is sunk solely for ventilation, custom usually dictates a size and shape consistent with the type commonly used for ordinary operations, because experience has demonstrated it can be excavated and maintained. In fact, most ventilation openings were never made with any consideration as to economic design for such use.

When an air course is required to pass air between different parts of mine workings under a fixed pressure, the size of opening required for a given volume of flow, is, of course, determined by the resistance factor of the air course. It is then impossible to proportion the opening with reference to economics of power consumption, and this paper is limited to the design of openings where ventilation pressure or power consumption may be varied for economic advantage.

As the size of an air course is increased, the expense involved in excavation and support is, of course, also increased, but the charge for power is decreased. In the case of a mine that has only a short assured operating life, it would not be economical to provide large openings in order to reduce power bills; but when operations are likely to continue for a long time, the power charge becomes a very important economic factor. Although lack of definite information on such factors as service life and costs of excavation and support affects the design of an air course, it appears, from a mathematical analysis of the problem, that considerable variations may be made in estimating these factors, without greatly

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altering the size of air course required for a given volume of flow. This results from the fact that all formulas, as to size of opening required for a given volume, include the factor (vol. in cu. ft. per min.),[3] which is so much greater than the other factors that it alone closely determines the economic size that should be used.

Mathematical Treatment

A mathematical treatment of the problem is given in the following pages. Inasmuch as exact formulas are too cumbersome for practical use, and minor variations in the different factors, other than volume, will have only slight effect on economic size, they may be eliminated. This, in effect, has been done by assuming approximately correct values for the factors, and a simplified equation has been developed for use which takes the form

$$\text{Clear area required, in sq. ft.} = Z\left(\frac{Q}{1{,}000}\right)^{\frac{6}{7}},$$

in which Z is a factor that varies with different types of mine openings, the value of which is given later, and Q represents volume.

The items of cost entering into this analysis of the design of airways are:

1. Cost of excavation;
2. Cost of support, such as concrete lining or timber, where necessary;
3. Annual power bill.

Assuming the life of the property is known (or may be estimated for purposes of calculation), the total cost of excavation and support may be distributed as an annual charge against operations. Ventilation power cost is generally considered a current expense; it will, of course, vary with the resistance of the individual air course. Economic design of the airway, therefore, calls for a minimum annual charge covering the total of these items.

Certain items of expense included in the general cost of ventilation have no bearing on the economic design of a primary air course, as, for instance, an outlet shaft or crosscut. For example, the power required to circulate air through the other mine workings is fixed for a given volume of flow; consequently, the charge for this power has no bearing on the design of the main air course. Similarly, the expense incident to the installation of fan equipment, will likewise have no bearing on the problem. Pressure requirements will vary with the proportions of the air course, affecting the fan design to a certain extent, but for practical purposes, the cost of installation may be considered as constant for a given volume of air.

Cost of air-course repairs is difficult to estimate, depending on ground conditions and character of support, but if likely to be a factor, it should be taken into account by adding a certain amount to the annual charge for excavation.

Calculation of Formulas

The formulas for the economic size of different type openings are calculated as follows. (Excavation outside primary limits is neglected throughout.):

Circular shaft or opening with no support or lining:

Clear area $= A = \pi r^2$;
frictional surface $= 2\pi r L$;
yearly charge for excavation $= \dfrac{Ce \times L \times \pi r^2}{N}$;

r = radius of shaft in ft.;
Ce = unit cost of excavation (or cost) per cu. ft.;
L = depth of shaft;
N = number of years assumed service life of opening.

Yearly charge for power $= \dfrac{Cp \times \text{air hp.}}{E}$,

where Cp = unit power cost per hp. year;
E = mechanical efficiency of fan installation based on volume and static pressure,

but air hp. $= \dfrac{\text{volume in c.f.m.} \times \text{pressure in lb. per sq. ft.}}{33{,}000}$,

and pressure $= \dfrac{K \times S \times V^2}{A}$,

in which K = coefficient of frictional resistance;
S = friction surface $= 2\pi r L$;
V = air velocity in ft. per min.;
A = clear cross-sectional area $= \pi r^2$;

velocity $= \dfrac{\text{volume}}{\text{area}}$;
volume $= Q$.

Therefore, air hp. $= \dfrac{K \times S \times Q^3}{33{,}000 \times A^3} = \dfrac{K \times 2\pi r L \times Q^3}{33{,}000 \times \pi^3 r^6}$;

yearly charge for power $= \dfrac{Cp \times K \times 2L \times Q^3}{E \times 33{,}000 \times \pi^2 r^5}$;

total yearly charge $= C$ = yearly charge for excavation plus yearly charge for power

$$= \frac{Ce \times L \times \pi}{N} r^2 + \frac{Cp \times K \times 2L}{E \times 33{,}000 \times \pi^2} \times \frac{Q^3}{r^5}.$$

To determine r for minimum charge C, differentiate C with respect to r.

$$\frac{dC}{dr} = \frac{2CeL\pi}{N}r - \frac{5Cp \times K \times 2L}{E \times 33{,}000 \times \pi^2} \times \frac{Q^3}{r^6}.$$

Placing this equal to zero and solving for r we have:

$$r^7 = \frac{5Cp \times K \times N \times Q^3}{Ce \times E \times 33{,}000 \times \pi^3}.$$

The formulas for other types of openings are deduced in the same manner. It is unnecessary to repeat the detailed mathematical steps, and only the final results follow.

Rectangular Shaft or Opening, with No Support.—The ratio of the longest side of the rectangle to the shortest side is designated as m. For a square cross-section m is, of course, equal to 1. The shortest side in this and all other rectangular openings is designated as a.

The final equation is:

$$a^7 = \frac{5 \times Cp \times K \times (1 + m) \times N \times Q^3}{E \times Ce \times 33{,}000 \times m^4}.$$

Circular Shaft or Opening, Supported by Concrete Lining.—The thickness of lining is generally determined by experience or judgment. It depends, however, upon the ground pressure and varies with the size of the shaft. For this analysis, it is assumed to be a direct function of r, the radius of the shaft inside support, and may be expressed as fr.

Unit cost of support, or cost per cu. ft. = Cs.

This unit cost is considered throughout as though the support were a continuous solid lining. Costs for an open framing of timber would, therefore, have to be reduced to this basis.

The final equation is:

$$r^7 = \frac{5 \times Cp \times K \times N \times Q^3}{E \times 33{,}000 \times \pi^3[Ce(1 + 2f + f^2) + Cs(2f + f^2)]}.$$

Rectangular Shaft, Supported by Concrete Lining or Timber.

The final equation is:

$$a^7 = \frac{5 \times Cp \times K \times (1 + m) \times N \times Q^3}{E \times 33{,}000 \times m^3[Ce(m + 2mf + 2f + 4f^2) + 2Cs(mf + f + 2f^2)]}.$$

Two-compartment Rectangular Shaft.—Centers, or dividing support assumed as being of same thickness as outside support, the final equation is:

$$a^7 = \frac{5 \times Cp \times K \times (1 + m) \times N}{E \times 33{,}000 \times m^3 \times 4[Ce(2m + 3mf + 4f + 6f^2) + Cs(3mf + 4f + 6f^2)]}.$$

Three-compartment Rectangular Shaft.—The final equation is:

$$a^7 = \frac{5 \times Cp \times K \times (1 + m) \times N \times Q^3}{E \times 33{,}000 \times m^3 \times [Ce(3m + 4mf + 6f + 8f^2) + 2Cs(2mf + 3f + 4f^2)]}.$$

Octagonal Shaft, Supported by Timber or Concrete.

$$r^{\prime} = \frac{5 \times Cp \times K \times N \times Q^3}{E \times 33{,}000 \times 3.31368^3[Ce(1 + 2f + f^2) + Cs(2f + f^2)]}$$

Rectangular Opening, Supported on Three Sides, Such as Timbered Cross-cut or Drift.

$$a^7 = \frac{5 \times Cp \times K \times (1 + m) \times N \times Q^3}{E \times 33{,}000 \times m^3[Ce(m + 2mf + f + 2f^2) + Cs(2mf + f + 2f^2)]}.$$

Variations in assumptions as to such factors as service life and the different cost units will, as has been previously stated, cause only slight differences in the size of air course required to carry economically a given volume. This is best demonstrated by comparing the size of air course, calculated from a definite set of conditions, with the size calculated for such other conditions as would exist if each of these factors, taken separately, were assumed to be either increased or decreased from the original figure. The basis of comparison is given in Table 1. It must be noted that the value of K that has been assumed, represents the lowest value practically obtainable with a nominally smooth surface. An unlined or timbered opening would have a much higher friction coefficient, so that the table should be used only for this purpose of comparison.

TABLE 1.—*Comparison of Different Types of Openings Designed for Equal Values of All Factors*

Assumed Value of Factors

$Vol.$ = 100,000 c.f.m.
N = 20-yr. service life
Cp = \$45.00 per hp. yr.
E = 0.65
K = 0.000,000,0015
Ce = \$0.35 per cu. ft.
Cs = \$1.50 per cu. ft.
m = 1.25
f = 0.20

Type of Opening	Radius r or Side a Lin. Ft.	Clear Area, Sq. Ft.	Area of Excavation, Sq. ft.
Unlined circular	4.34	59.2	59.2
Unlined rectangular	7.01	61.4	61.4
Lined circular	3.65	41.8	60.3
Lined rectangular	5.50	37.8	70.0
2-compartment rectangular	4.16	43.2	74.2
3-compartment rectangular	3.52	46.5	77.7
Octagonal, lined	3.57	42.3	60.8
Rectangular supported on three sides	5.69	40.6	65.6

TABLE 2.—*Effect on Sizes of Shaft Due to Variations in Assumed Service Life*

All Other Factors Same as in Preceding Table

Type of Opening	N − 10 years		N − 20 years		N − 40 years	
	Radius r or Side a, Ft.	Clear Area, Sq. Ft.	Radius r or Side a, Ft.	Clear Area, Sq. Ft.	Radius r or Side a, Ft.	Clear Area, Sq. Ft.
Unlined circular	3.93	48.5	4.34	59.2	4.79	72.1
Unlined rectangular	6.35	50.4	7.01	61.4	7.74	74.9
Lined circular	3.31	33.4	3.65	41.8	4.06	51.2
Lined rectangular	4.98	31.0	5.50	37.8	6.07	46.0
2-compartment rectangular	3.77	35.5	4.16	43.2	4.60	53.0
3-compartment rectangular	3.19	38.2	3.52	46.5	3.89	56.8
Octagonal	3.26	34.8	3.59	42.3	3.97	51.7
Rectangular—supported on 3 sides	5.15	33.2	5.69	40.6	6.28	49.3

A variation of 100 per cent. in assumed service life of opening, therefore, makes a difference of approximately 10 per cent. in length of radius or side, and a difference of approximately 20 per cent. in the clear area. It is further evident from an examination of the formulas that variations in the cost of power and friction factor will have exactly the same proportional effect on size of opening.

TABLE 3.—*Effect of Size of Shaft Due to Variations in Cost of Excavation*

Type of Opening	C_e − 0.175		C_e − 0.35		C_e − 0.70	
	Radius r or Side a, Ft.	Clear Area, Sq. Ft.	Radius r or Side a, Ft.	Clear Area, Sq. Ft.	Radius r or Side a, Ft.	Clear Area, Sq. Ft.
Unlined circular	4.79	72.0	4.34	59.2	3.93	48.5
Unlined rectangular	7.74	75.0	7.01	61.4	6.35	50.4
Lined circular	3.78	45.0	3.65	41.8	3.48	38.0
Lined rectangular	5.64	39.6	5.50	37.8	5.27	34.7
2-compartment rectangular	4.30	46.2	4.16	43.2	4.00	40.0
3-compartment rectangular	3.63	49.4	3.52	46.5	3.37	42.6
Octagonal	3.72	45.8	3.59	42.3	3.44	39.2
Rectangular—supported on 3 sides	5.86	43.0	5.70	40.6	5.44	37.0

A variation of 100 per cent. in assumed unit cost of excavation causes a difference of approximately 4 per cent. in radius or side of lined shafts, and about 10 per cent. difference in the unsupported openings.

TABLE 4.—*Effect of Size of Shaft Due to Variations in Unit Cost of Lining or Support*

Type of Opening	$C_s = 0.75$		$C_s = 1.50$		$C_s = 3.00$	
	Radius r or Side a, Ft.	Clear Area, Sq. Ft.	Radius r or Side a, Ft.	Clear Area, Sq. Ft.	Radius r or Side a, Ft.	Clear Area, Sq. Ft.
Lined circular	3.82	46.0	3.65	41.8	3.44	37.2
Lined rectangular	5.82	42.3	5.50	37.8	5.11	32.6
2-compartment rectangular	4.40	48.4	4.16	43.2	3.88	37.6
3-compartment rectangular	3.72	52.0	3.52	46.5	3.28	40.4
Octagonal	3.80	47.8	3.59	42.3	3.37	37.6
Rectangular—supported on 3 sides	6.00	45.0	5.69	40.6	5.30	35.2

DEVELOPMENT OF SIMPLIFIED FORMULAS

The further mathematical treatment of this subject now depends upon practical considerations of application. Many refinements are necessary to absolute accuracy. However, the theoretical formulas that have been developed are entirely too cumbersome for ordinary use, and it does not seem necessary, for practical purposes, to adopt an absolutely accurate form. A definite assumption as to the productive life of a property is seldom warranted, and all other factors affecting design will vary to some extent from any preliminary estimate. Although variations in the unit cost of excavation will greatly affect the total cost of a shaft, it has been shown that the effect on economic size is relatively light, and, in general, minor variations in any of the factors other than volume are also negligible. It seems, therefore, as has been previously stated that approximately accurate values of the different factors may be substituted in the theoretical formulas and a simplified formula then be developed. The proposed formula, or equation, takes the form:

$$\text{Clear area} = Z\left(\frac{Q}{1{,}000}\right)^{\frac{6}{7}},$$

in which Z is a factor that is variable for different types of mine openings. Its derivation, in the case of a circular, unsupported opening, is as follows:

$$\text{Clear area} = \pi r^2.$$

$$\text{For economic size, } r = \left(\frac{5 \times Cp \times K \times N \times Q^3}{Ce \times E \times 33{,}000 \times \pi^3}\right)^{\frac{1}{7}};$$

$$r^2 = \left(\frac{5 \times Cp \times K \times N \times 1{,}000^3}{Ce \times E \times 33{,}000 \times \pi^3}\right)^{\frac{2}{7}} \times \left(\frac{Q}{1{,}000}\right)^{\frac{6}{7}};$$

$$\text{clear area} = \pi r^2 = \pi\left(\frac{5 \times Cp \times K \times N \times 1{,}000^3}{Ce \times E \times 33{,}000 \times \pi^3}\right)^{\frac{2}{7}}\left(\frac{Q}{1{,}000}\right)^{\frac{6}{7}}.$$

$$\text{Placing } Z = \pi\left(\frac{5 \times Cp \times K \times N \times 1{,}000^3}{Ce \times E \times 33{,}000 \times \pi^3}\right)^{\frac{2}{7}} = \text{ we have,}$$

$$\text{clear area} = Z\left(\frac{Q}{1{,}000}\right)^{\frac{6}{7}}.$$

The proposed equation indicates that for a given volume of flow, the economic area is closely constant over a considerable variation of conditions. This, in brief, results from the fact that power required to force air through a duct varies as the cube of the volume, and from the other relationships of pressure, velocity, and area that are well known.

Values of Z for the more common types of mine airways are given in Table 5. The values of K were taken from U. S. Bureau of Mines *Reports of Investigations* No. 2681, "Friction Factors for Metal Mine Airways." Volumes of all other factors given are those used in calculating the value of Z in each case.

TABLE 5.—*Simplified Formulas for Economic Size of Openings Required Where Ventilating Pressure Is Not Fixed*

$$\text{Clear Area} = Z\left(\frac{Q}{1000}\right)^{\frac{6}{7}}$$

Type of Opening	Assumed Values of Factors			Value of Z
	C_s, Per Cu. Ft.	C_e, Per Cu. Ft.	K	
Unlined circular (sedimentary rock)		$0.35	0.000,000,006	1.74
Unlined circular (blocky igneous rock)		0.35	0.000,000,018	2.35
Unlined rectangular (sedimentary rock)		0.35	0.000,000,006	1.76
Unlined rectangular (blocky igneous rock)		0.35	0.000,000,018	2.40
Lined circular	$1.5	0.35	0.000,000,002	0.88
Lined rectangular	1.5	0.35	0.000,000,002	0.80
Rectangular—supported on 3 sides by concrete	1.5	0.20	0.000,000,002	0.90
Rectangular—supported on 3 sides by open timber	0.4	0.20	0.000,000,009	1.75
2-compartment rectangular—concrete lined	1.5	0.35	0.000,000,002	0.92
2-compartment rectangular—open timber	0.4	0.35	0.000,000,009	1.68
3-compartment rectangular—concrete lined	1.5	0.35	0.000,000,002	0.98
3-compartment rectangular—open timber	0.4	0.35	0.000,000,009	1.78
Octagonal—skin-to-skin timber	0.75	0.35	0.000,000,002	1.00

In all cases N = 20 yr.; Cp = $45.00 per hp. yr.; E = 0.65; m = 1.25; f = 0.2.

The sizes of openings that are required to carry a flow of 100,000 c.f.m. are given in Table 6, together with the annual costs for an air course 1,000 ft. long. The basis of costs is the unit charges that have been assumed for the simplified formulas. This table is offered only for the purpose of comparing the different types of air courses.

TABLE 6.—*Comparison of Different Types of Air Courses*

[For an air course 1,000 ft. long to carry 100,000 c. f. m.]

Type of Openings	Dimensions: Side a or Radius r, Lin. Ft.	Dimensions: Clear Area, Sq. Ft.	Dimensions: Area of Excavation, Sq. ft.	Annual Expense Items: Excavation Costs	Annual Expense Items: Support Costs	Annual Expense Items: Excavation and Support Costs	Annual Expense Items: Power Costs	Annual Expense Items: Total Costs
Unlined circular (sedimentary rock)......	5.35	90.00	90.00	$1,575		$1,575	$ 580	$2,155
Unlined circular (igneous blocky rock)....	6.18	120.30	120.30	2,105		2,105	846	2,951
Unlined rectangular (sedimentary rock)...	8.54	91.22	91.22	1,596		1,596	958	2,554
Unlined rectangular (igneous blocky rock)	10.00	125.00	125.00	2,187		2,187	1,305	3,492
Lined circular........................	3.80	45.56	65.20	1,140	$1,475	2,615	1,079	3,684
Rectangular (concrete lined)............	5.72	40.97	75.50	1,321	2,590	3,911	1,579	5,490
Rectangular (framed timber support).....	7.80	76.20	140.80	2,462	1,292	3,754	1,506	5,260
2-compartment rect. (concrete lined).....	4.34	47.08	80.70	1,410	2,520	3,930	1,655	5,585
2-compartment rect. (framed timber support)............................	5.88	86.64	148.30	2,610	1,233	3,243	1,432	5,275
3-compartment rect. (concrete lined).....	3.66	50.55	84.00	1,470	2,510	3,980	1,636	5,616
3-compartment rect. (framed timber support)............................	4.96	92.10	154.00	2,698	1,238	3,936	1,624	5,560
Rectangular supported on (3 sides concrete)	6.06	46.19	74.60	746	2,210	2,956	1,183	4,139
Rectangular (supported on 3 sides)......	8.42	88.84	144.00	1,440	1,103	2,543	1,029	3,572
Octagonal skin-to-skin timber...........	3.96	51.94	74.50	1,305	846	2,151	760	2,911

Conclusions

1. In ground, such as sedimentary rock, or coal that breaks in such a way as to leave an even surface and, therefore, has a low factor of resistance to the flow of air, an unlined or unsupported opening is most economical. This, of course, is common coal-mine practice wherever possible.

2. In igneous, blocky ground, which has a very high factor of resistance, it is not much cheaper to drive an unsupported opening than it is to drive an opening requiring concrete or timber lining. This is due to the fact that the resistance to flow of air is lower with both concrete or timber support, and a smaller-sized opening may be used. For the same reason, a concrete lining will, under any circumstances, not be much more expensive than timber and under certain conditions may be more economical, because repair costs are likely to be considerably lower. To a certain extent all figures given in the tabulation are affected by the assumption that thickness of lining, or support, varies with the size of opening.

3. In the case of rectangular supported openings, no great increase in size or cost occurs when the number of parts, into which the total area is divided, is increased. This, again, is due to some extent to the method of figuring the size of support, and its bearing on excavation. However, two- or three-compartment shafts are structurally much stronger than the single-compartment shaft of same air-carrying capacity, and there seems to be no economic argument against their use where shafts of this type are to be adopted.

4. The established preference for the circular concrete-lined shaft for ventilation purposes is shown to be well founded in economics. Where the lowest initial cost is the primary consideration, the similar octagonal shaft with skin-to-skin timber has marked advantages over other types. In this and all other cases involving a comparison of concrete and timber, the relative life of these materials needs to be considered. It is doubtful whether timber will last 20 years as has been assumed in all cases.

5. In the case of mines which have a long period of productive activity, the tendency has most commonly been to make the air courses too small rather than too large. A considerable amount of the money expended in power bills might have been easily saved by an economic design.

Permissible Limits of Toxic and Noxious Gases in Mine and Tunnel Ventilation*

By R. R. Sayers,† Washington, D. C.

(New York Meeting, February, 1926)

Ventilation may be defined as the process by which vitiated air of an enclosed or partly enclosed space is continuously replaced by fresh air. Fresh air has been defined as invigorating pure air. Pure dry air at sea level contains the following gases:[1]

Analysis of Air at Sea Level

	Per Cent.
Oxygen	20.94
Nitrogen	78.09
Carbon dioxide	0.03
Argon	0.94
Helium, krypton, neon, xenon, hydrogen, hydrogen peroxide, ammonia. ozone	traces

Carbon Dioxide Limit

For many years carbon dioxide was used as an index of the purity of the air vitiated by the metabolism of people. It was, however, well known that the untoward effects produced on those exposed to such air was not due to the concentration of the carbon dioxide. It has been found that the untoward effects on the comfort and efficiency of those so exposed are due to the increased temperature and humidity. It has been found that men can breathe air containing many times the amount of carbon dioxide found in our worst ventilated theaters and assembly halls, which, according to Rosenau, do not contain above 0.5 per cent. carbon dioxide.

Nevertheless, in mines it sometimes occurs in sufficient quantities to cause symptoms in men or even unconsciousness and death. One-half of 1 per cent. of carbon dioxide in normal air causes a slight and unnoticeable increase in the ventilation of the lungs, that is, a man exposed to one-half of 1 per cent. of carbon dioxide will breathe a little deeper and a little faster than when in pure air. If 2 per cent. of carbon dioxide is in the air, the lung ventilation will be increased about 50 per cent.; if there

* Published with the approval of the Director, U. S. Bureau of Mines.

† Chief Surgeon, U. S. Bureau of Mines; Surgeon, U. S. Public Health Service.

[1] M. J. Rosenau: Preventive Medicine and Hygiene. (3rd Ed., 1920), 662.

is 3 per cent., the lung ventilation will be increased about 100 per cent.; 5 per cent. causes about 300 per cent. increase in the lung ventilation and the breathing is laborious; 10 per cent. cannot be endured for more than a very few minutes.

These results are obtained from carbon dioxide air when the oxygen in the air remains about normal and the subject is not working. According to Sollmann, if oxygen deficiency is excluded by inhaling gas mixtures containing 20 per cent. of oxygen, no effects occur until the concentration of 3 per cent., volume, of carbon dioxide is reached. With this concentration, there is some hyperpnea and discomfort: 8½ per cent. produced distinct dyspnea in man in a few minutes, with rise of blood pressure and congestion, which become insupportable in 15 or 20 min. but disappear promptly in fresh air. The symptoms increase to 15 per cent.; but even 20 per cent. is not dangerous in an hour to animals, and probably not to man.

With 25 to 30 per cent., the stimulant phenomena pass into depression, with diminished respiration, fall of blood pressure, coma (generally without convulsions), loss of reflexes, anesthesia, and gradually death after some hours, the heart outlasting the respiration. With higher concentrations, the stimulation is still briefer. With pure carbon dioxide, death may occur in a few minutes as a mixed effect of carbon dioxide and anoxemia.

Oxygen Required

Oxygen is the most important to man of the gases in the air. Although this gas is not usually considered as toxic or noxious, a variation in its concentration cannot be neglected, as untoward effects develop if the variation is marked. Man is so made that he breathes easily and works best when the air contains about 21 per cent. of oxygen, which is the amount usually in air. Yet he is able to live and work, though not so well, when there is less oxygen. If about 17 per cent. of the air is oxygen, a man at work will breathe a little faster and a little deeper, about the same as when he first goes from sea level to a height of 5000 ft.

Men breathing air that has as little as 15 per cent. of oxygen usually become dizzy, notice a buzzing in the ears, have a rapid heartbeat and often suffer from headache. Very few men are free from these symptoms when the oxygen in the air falls to 10 per cent. Haldane, the English physiologist, says that under certain conditions men may be conscious even with as little as 3½ per cent. oxygen in the air they are breathing. However, under other conditions, men faint, or become unconscious, when the air contains 9 per cent. oxygen or more.

The flame of a candle or Wolf lamp goes out when the air has 16½ per cent. oxygen or less, and the flame of a carbide or acetylene lamp goes out when there is 13 per cent. or less. Although a man does not usually

lose consciousness until very much less than 13 per cent. oxygen is present, no one should try to enter a mine in which a flame will not burn unless he wears a self-contained oxygen breathing apparatus.

Excess Oxygen

The effects of breathing oxygen in concentrations above normal have been studied by many investigators, beginning as early as 1774 when Priestley rediscovered the gas. Some of the early observers reported that breathing pure oxygen irritated the lungs. Professor Lorrain Smith[2] showed that the irritating effects of oxygen are to be found only after 48 hr. of continuous exposure to an atmosphere containing over 80 per cent. O_2. Dr. Leonard Hill[2] pointed out that mules were exposed to an oxygen concentration of 60 per cent. for more than a year in the Hudson tunnels and they remained in perfect health. Rats exposed to 98.5 per cent. oxygen die as a result of lung irritation after about 72 hr.

Oxygen causes death rapidly when the pressure is increased 2 to 4 atmospheres, or 300 to 400 per cent. atmosphere oxygen.[3] Lorrain Smith reported that two larks at 300 per cent. atmosphere O_2 had violent convulsions in 13 min. Paul Bert found that convulsions appeared in mice after 12 to 30 min. exposure to oxygen at 4 to 5 atmospheres pressure.

Pure oxygen at 1, 2, 3, or 4 atmospheres has about the same effect as air at 5, 10, 15, or 20 atmospheres. Dr. Schereschewsky in an article on compressed-air illness states: "In the salvaging operations of the submarine F-4, a diver, W. F. L., became entangled and remained for a total time of 3 hr. and 45 min. below the surface, at an average depth of 250 ft. The high partial pressure of oxygen to which this diver was subjected resulted in bronchopneumonia, which, according to French, was some time in clearing up."

Control of Amount of Oxygen

By controlling the amount of oxygen present at any pressure, ill effects can be avoided. As an example, if the oxygen present is reduced to 5 per cent. at 1 atmosphere, a pressure of slightly more than 4 atmospheres is needed to bring the equivalent partial pressure of oxygen to normal, 20.9 per cent., and the concentration produced by 10 or 12 atmospheres would cause no harmful effects due to oxygen.

Limits of Other Gases

Effects of variations in the amounts of the physiologically active gases that are normally found in air have been briefly described, nitrogen

[2] Leonard Hill: Caisson Sickness and the Physiology of Work in Compressed Air. (London, 1912) 255 pages.

[3] Pure oxygen at one atmosphere would have a partial pressure of 14.7 lb. but in much of the literature on the partial pressure of gases it is referred to in percentages, *i.e.*, pure oxygen is referred to as 100 per cent. at 1 atmosphere pressure, 200 per cent. at 2 atmospheres pressure, etc.

being physiologically inactive. The effects of gases abnormally found in the air of mines and tunnels may be far more important.

Methane, or marsh gas, is especially important in coal mines and may be present occasionally in metal mines and in tunnels. Its importance is not due to its physiological or noxious action, but to the fact that it forms explosive mixtures with the oxygen of the air and this may result in disaster. Furthermore, the methane may dilute the oxygen of the air to such an extent as to produce the effects of low oxygen mentioned above.

Sulfur dioxide in mines usually comes from the decomposition of sulfide minerals or from burning of explosives that contain sulfur. It is very irritating to the eyes and respiratory passages, one part in 500 being almost intolerable to breathe; occasionally the concentration in the mine atmosphere is enough to be dangerous. It is easily recognized by its characteristic odor and causes choking when breathed, as do fumes from burning sulfur.

Oxides of nitrogen are given off when nitroglycerine, blasting gelatin, dynamite or other high explosives are burned rather than exploded. Haldane found that when nitroglycerine was exploded, the gases of combustion were 63.2 per cent. carbon dioxide and 31.6 per cent. nitrogen; but when burned in the presence of its own gases they contained 35.9 per cent. carbon monoxide and 48.2 per cent. nitric oxide (NO). This latter gas on contact with air is immediately oxidized to peroxide of nitrogen with the production of red fumes.

Peroxide of nitrogen is probably the most irritating of any of the oxides of nitrogen. The effects on the respiratory passages do not usually manifest themselves for several hours after exposure, when edema and swelling take place. This irritation may be followed by bronchitis or pneumonia.

Hydrogen sulfide (stink damp) is sometimes found in mine air and in caissons, but usually in small amounts. In low concentrations, it has a very repulsive odor which may serve as a warning. It is produced in mines in high sulfide ores by some types of decomposition of such ores and under some conditions from the use of explosives in this type of ore. In the sinking of caissons in the construction of bridge piers, hydrogen sulfide is sometimes encountered. It has sometimes been attributed to the decomposition of organic debris.

Hydrogen sulfide is very poisonous. Wherever it exists the possibility of poisoning is present. Its toxicity is comparable to that of hydrogen cyanide. Poisoning by hydrogen sulfide is of two types, acute and subacute—causing asphyxiation and irritation (conjunctivitis, bronchitis, pharyngitis and depression of the central nervous system), respectively. Death from asphyxia is caused by paralysis of the respiratory center, whereas death from subacute poisoning is associated with edema of the lungs.

The exact low limit of hydrogen-sulfide concentration at which it ceases to act as a poison has not as yet been determined, but is evidently below 0.005 per cent. From 0.06 to 0.1 per cent. is enough to cause serious symptoms within a few minutes. In low concentrations, hydrogen sulfide produces headache, sleeplessness, dullness, dizziness and weariness. Pain in the eyes, followed by conjunctivitis, is fairly constant; bronchitis and pains in the chest are frequent. Further poisoning produces depression, stupor, unconsciousness and death. The heart continues to beat after respiration has ceased.

Sources and Effects of Carbon Monoxide

The presence of carbon monoxide in the air of mines and tunnels probably is the most frequent cause of poisoning in such places. Carbon monoxide is produced by the burning of carbon-containing materials, as coal, wood, gasoline and explosives, whenever the supply of air or oxygen is not sufficient for complete combustion.

The poisoning, as usually encountered, is an acute condition resulting from the breathing of an atmosphere containing carbon monoxide. A number of cases described in the literature show that carbon monoxide was often a cause of death by accident or suicide, and a means of punishment or torture. Aristotle, who lived 384–322 B. C., stated that "animals collapse from harmful odors, as man gets a severe headache and often dies through charcoal vapors." The occurrence of carbon monoxide poisoning has increased in frequency through the years; it has occurred in homes and in many of the industries. The following table gives the approximate percentage of carbon monoxide found in gases from various sources of poisoning:[4]

Sources of Carbon Monoxide Poisoning

	CO by Volume, Per Cent.
Mine explosion, immediately after dust explosion (experimental)	8.0
Mine explosion, 1 day after explosion in coal mine	1.0
Mine fire	1.0
Blasting with 40 per cent. gelatin dynamite, 7 min. after shooting 100 sticks	1.2
Blasting:	
Products of combustion of black blasting powder	10.8
Products of combustion of 40 per cent. nitroglycerine dynamite	28.0
Products of combustion of 40 per cent. ammonia dynamite	5.0
TNT	60.0
Automobile exhaust gas (average of tests on 101 cars of all types)	7.0
Railroad locomotive stack-gas	2.0

[4] Analyses obtained from the Bureau of Mines gas laboratory at Pittsburgh, Pa.

From the above analyses it is seen that carbon monoxide occurs often in mines and is found in railroad tunnels and in vehicular tunnels used by automobiles. It was due to this last fact that the New York State Bridge and Tunnel Commission and the New Jersey Interstate Bridge and Tunnel Commission requested the U. S. Bureau of Mines to make an investigation of carbon monoxide and to determine the maximum permissible limit for tunnel air. This work was begun by Dr. Yandell Henderson, Consulting Physiologist of the U. S. Bureau of Mines, and was later confirmed and extended by studies at the Bureau's station at Pittsburgh and at the experimental mine at Bruceton, where a tunnel was constructed in the mine about one quarter the size of the tubes to be built for the vehicular tunnel under the Hudson River.

Action of Carbon Monoxide in the Blood

Carbon monoxide exerts its extremely dangerous action on the body by displacing the oxygen from combination with the hemoglobin. Hemoglobin, the coloring matter of the blood, normally absorbs oxygen from the air and delivers it to the tissues through the blood. The affinity of carbon monoxide for hemoglobin is about 300 times that of oxygen. Therefore, even when only a small amount of the poisonous gas is present in the air breathed into the lungs, much of the hemoglobin is locked up in combination with carbon monoxide and so cannot keep up its usual work of carrying oxygen to the tissues, which, because of the lack of oxygen, cannot do their work properly. If the tissue cells are smothered long enough they become damaged, and the injury to the cells may be permanent even if the patient survives.

It has been asserted that carbon monoxide has a specific poisonous action on some tissues of the body, especially those of the nervous system, but there is little evidence in favor of this statement and much against it. Haggard and Henderson found that there was no change in the rate of growth of chick brain tissue, even when it was exposed to an atmosphere containing over 70 per cent. of carbon monoxide. It has been shown many times that animals without red blood (hemoglobin) can live in atmospheres containing high concentrations without apparent harmful effects.

With increasing concentrations of carbon monoxide, the time required for a given amount of hemoglobin to combine with carbon monoxide would decrease very rapidly, until with 1.0 per cent. only a few breaths may produce a saturation of 60 to 80 per cent., which is fatal. For a person at rest, it can be assumed that approximately 80 per cent. of the equilibrium values is attained after the following periods of time:

Time in Which Different Amounts of CO in Air Will Saturate the Breather's Blood

Concentration of Carbon Monoxide in Air (Inclusive), Per Cent.	Blood Saturation (80 Per Cent. of Approximate Equilibrium Values), Per Cent.	Time, Hr.
0.02 to 0.03	23 to 30	5 to 6
0.04 to 0.06	36 to 44	4 to 5
0.07 to 0.10	47 to 53	3 to 4
0.11 to 0.15	55 to 60	1½ to 3
0.16 to 0.20	61 to 64	1 to 1½
0.20 to 0.30	64 to 68	½ to ¾
0.30 to 0.50	68 to 73	[a]20 to 30
0.50 to 1.00	73 to 76	[a] 2 to 15

[a] Minutes.

Symptoms

The symptoms of carbon-monoxide poisoning may be divided into two stages, the first covering the period beginning with normal and ending in syncope, and the second a depression of the central nervous system beginning in syncope, extending through coma and ending in apnea.

Stage 1.—Tightness across forehead, dilatation of cutaneous vessels, headache (frontal and basal), throbbing in temples, weariness, weakness, dizziness, nausea and vomiting, loss of strength and muscular control, increased pulse and respiratory rates, and collapse. All of these are greatly increased and accelerated with exercise because of the additional need of oxygen in the tissues. Men at rest have often been exposed to carbon monoxide all day without noticing any marked ill-effects, but on walking home or exercising have experienced severe symptoms, even to unconsciousness.

It is seldom that all of these symptoms are experienced by the same individual. Also, in some persons, the poisoning may proceed to the stage of syncope without the victim feeling any of the subjective symptoms, this often occurring if the poisoning has been rapid.

Stage 2.—Increased pulse and respiratory rates, fall of blood pressure, loss of muscular control especially sphincters, loss of reflexes, coma usually with intermittent convulsions, Cheyne-Stokes's respiration, slowing of pulse, respiration slow and shallow, cessation of respiration, death.

With a given blood saturation, the character and severity of symptoms acquired during exposure depend upon the time required to attain that saturation and the degree of muscular activity—in other words, the extent of oxygen deprivation. The number of symptoms decreases with the rate of saturation. With high concentrations the victim may experience but few (weakness and dizziness) of those given under stage 1.

If a given saturation has been acquired by a long exposure to a low concentration, the symptoms and after effects will be much more severe when a given saturation has been acquired than by a short exposure to a high concentration. Muscular activity increases the number and accentuates the character of the symptoms during exposure, and will bring out latent symptoms after exposure. A person at rest may pass into a state of dizziness and unconsciousness without experiencing any marked previous effects.

In general, the predominating symptoms accompanying the various percentage saturations are as follows:

Symptoms at Different Concentrations of CO

Percentage of Blood Saturation	Symptoms
0–10	No symptoms.
10–20	Tightness across forehead; possibly slight headache, dilatation of cutaneous blood vessels.
20–30	Headache; throbbing in temples.
30–40	Severe headache, weakness, dizziness, dimness of vision, nausea and vomiting, collapse.
40–50	Same as above with more possibility of collapse and syncope, increased respiration and pulse.
50–60	Syncope, increased respiration and pulse; coma with intermittent convulsions; Cheyne-Stokes's respiration.
60–70	Coma with intermittent convulsions, depressed heart action and respiration, possibly death.
70–80	Weak pulse and slowed respiration; respiratory failure and death.

After Effects

The after effects of carbon monoxide poisoning range from practically none to headaches, muscular pains, long periods of unconsciousness, loss of strength, mental derangement such as loss of memory, paralysis and temporary blindness. In nearly all cases these clear up in a day or two, but in some cases they have extended over several months, and, in a few instances, over years.

The possibility and severity of these effects depend upon the duration of oxygen deprivation, personal idiosyncrasy, and physical condition before exposure. After coal-mine explosions it is a common occurrence to find groups of men, some dead and others not unconscious, all of whom have been exposed to the same concentrations of gas for the same period of time.

Slight exposures to high concentrations cause far less tissue damage than long exposures to low concentrations, the effect of oxygen deprivation being additive. With some men, there is a specific tolerance to this deprivation, and by this they may experience little or no after effects from exposures which are serious to fellow workmen. It is very probable

that much of this idiosyncrasy is due to previous health condition, and that many diseases, as cerebral arterio-sclerosis, syphilis, and acute alcoholism, are contributory to serious after effects.

Carbon Monoxide Limits

At the conclusion of his studies, Dr. Henderson advised that if the Hudson vehicular tunnel were so constructed that persons passing through would be exposed to not more than 4 parts of carbon monoxide in 10,000 parts of air (0.04 per cent.) for not longer than 45 min., they would experience no ill effects. The Bureau of Mines confirmed these findings and continued its studies on the effects of long exposures to low concentrations, of strenuous exercise, of high temperatures and humidities in low concentrations. Following is a summary of the Bureau's results:

Relation of Length of Exposure, Exercise, Temperature, Humidity and Degree of Concentration of CO

With the subject at rest:

Exposure for 6 hr. to 2 parts of CO in 10,000 caused

a. Saturation of 16 to 20 per cent. of the hemoglobin of the blood with CO.
b. Very mild subjective symptoms of CO poisoning at the end of the test.
c. No noticeable effects after the test.

Exposure to 3 parts of CO caused

a. Saturation of 22 to 24 per cent. of the hemoglobin with CO after 4 hr., and 26 to 27 per cent. after 5 hr.
b. Symptoms at the end of 2 hr. absent; after 4 hr., mild effects attributed to CO poisoning; and after 5 hr., moderate effects.
c. After effects of 4 hr. exposure, mild; of 5 hr. exposure, moderate.

Exposure to 4 parts of CO in 10,000 caused

a. Saturation of 15 to 19 per cent. of the hemoglobin with CO at the end of 1 hr., and 21 to 28 per cent. at the end of 2 hr.
b. After effects, moderate to marked.

With the subject exercising strenuously:

Exposure for 1 hr. to 2½ parts of CO in 10,000 caused

a. Saturation of 14 to 16 per cent. of the hemoglobin with CO.
b. Moderate symptoms of CO poisoning at the end of the test.
c. After effects, mild to moderate.

Exposure for 1 hr. to 3.3 parts of CO in 10,000 caused

a. Saturation of 17 per cent. of the hemoglobin with CO.
b. Mild to moderate symptoms of CO poisoning.
c. After effects, mild to moderate.

Exposure for 1 hr. to 4 parts of CO in 10,000 caused

a. Saturation of 23 per cent. of the hemoglobin with CO.
b. Moderate symptoms of CO poisoning.
c. Moderate after effects.

With the subject at rest, temperature and humidity high:

Exposure for 1 hr. to 3.1 parts of CO in 10,000 caused

a. Saturation of 16 per cent. of the hemoglobin with CO.
b. Mild symptoms of CO poisoning.
c. Mild to moderate after effects.

Test for Carbon Monoxide Poisoning

A diagnosis of carbon monoxide poisoning is usually made from a correlation of the history and place of possible exposure with the symptoms produced. But this is not always positive evidence, because carbon monoxide exists at unsuspected places, and the symptoms produced are common to other causes. The only infallible test is by examining the blood for carbon monoxide hemoglobin, for which many methods have been devised.

The Bureau of Mines has developed the pyrotannic acid method for this test. By its use a small amount of blood (0.10 c. c., which can be procured from a puncture wound in the finger) can be quantitatively examined in a few minutes for carbon monoxide and a true diagnosis made. The apparatus is portable (pocket size), and the technique for use is sufficiently easy for unskilled users to make an accurate diagnosis, as well as to determine later when the carbon monoxide has been removed.

Prevention of Poisoning from Gases

The chief factors in preventing poisoning by the various toxic or noxious gases that may occur in mines or tunnels are:

1. Good ventilation, as this will dilute and carry away the gases.
2. Avoidance as much as possible of any exposure to air known to contain poisonous gases.
3. Use of adequate protective equipment when atmospheres known to contain poisonous gases are to be encountered; to keep calm when necessary to enter or be exposed to unprotected atmospheres containing gases; not to hurry but to get to fresh air as quickly but with as *little exertion* as possible.

Treatment for Persons Poisoned

The steps in effective treatment of acute poisoning by poisonous or noxious gases in mine or tunnel air are:

1. Removal of the victim to fresh air as soon as possible.
2. Restoration of normal breathing. If breathing has stopped, or is weak and intermittent, or present in but occasional gasps, artificial respiration by the Schaefer method should be given persistently until normal breathing is resumed, or until after the heart has stopped.
3. Restoration of circulation aided by rubbing the limbs and keeping the body warm with blankets, hot-water bottles, hot bricks or other devices, care being taken that these are wrapped or do not come in contact with the body and produce burns. This aids in tiding the body over a period of low vitality. Other stimulants, such as hypodermics of caffein, sodium benzoate, or camphor in oil, should not be administered except by a doctor after he has considered the possibility of over-stimulation and consequent collapse.

4. Rest for the patient who should be kept lying down in order to avoid any strain on the heart. Later, he should be treated as a convalescent and given plenty of time to rest and recuperate.

5. Symptomatic treatment of after effects of poisoning by such gases.

6. It should be emphasized that inhalations, for 20 to 30 min., of oxygen, or a 5 per cent. mixture of carbon dioxide in oxygen if available, will when *given immediately* greatly lessen the number and severity of symptoms of carbon monoxide poisoning, as well as decrease the possibility of serious after effects. All industries in which this type of poisoning commonly exists should provide apparatus (inhalers) for the efficient administration of these treatments. This apparatus should be placed at places most convenient for treating carbon monoxide poisoning, and employees should be trained in its use so that resuscitation may be effected immediately.

DISCUSSION

Carbon Monoxide Recorder

A. C. Fieldner,* Pittsburgh, Pa. (written discussion).—It may be of interest, in connection with the permissible limit of carbon monoxide in

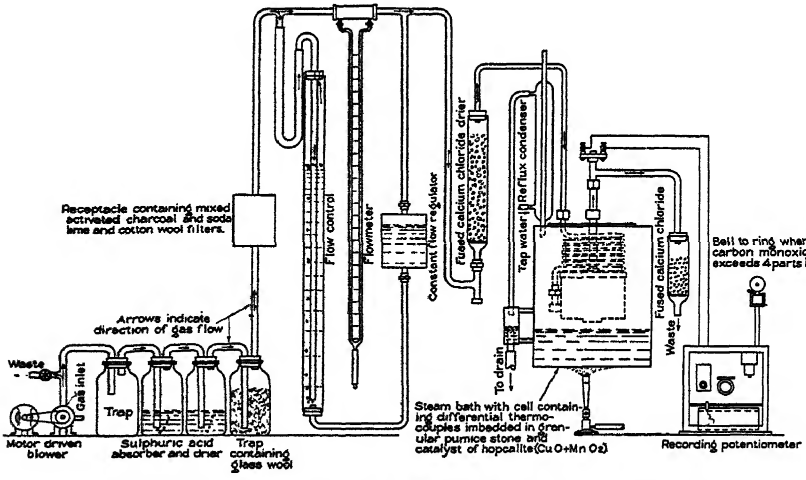

Fig. 1.—Essential parts of carbon monoxide recorder.

ventilated tunnels, to call attention to the very sensitive analyzing and recording apparatus for carbon monoxide that has been developed at the Pittsburgh Experiment Station of the Bureau of Mines. The need for such an instrument to control ventilation in tunnels and mines led the Bureau to undertake this problem with the object of devising a satis-

* Chief Chemist, U. S. Bureau of Mines and Superintendent Pittsburgh Experiment Station, Pittsburgh, Pa.

factory apparatus that should indicate and record carbon monoxide in air to a sensitivity of 1 part CO in 100,000 parts of air.

The principle employed is selective combustion of the carbon monoxide in the presence of a catalytic material in which the temperature rise is

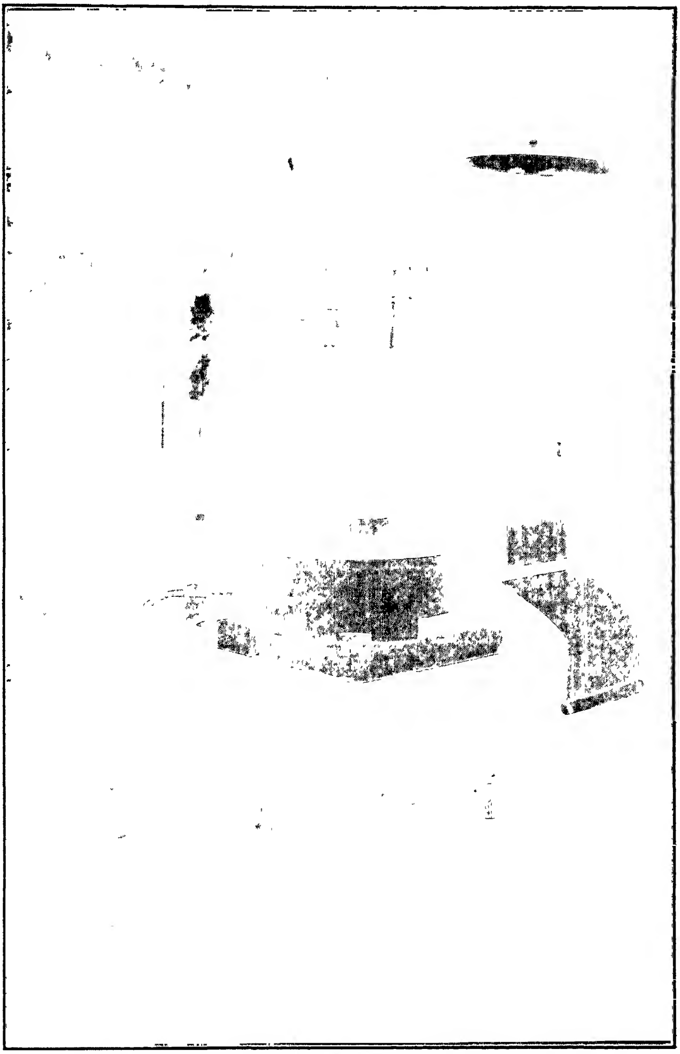

FIG. 2.—AUTOMATIC CARBON MONOXIDE RECORDER INSTALLED IN FAN HOUSE OF LIBERTY TUNNELS, PITTSBURGH, PA.

measured by differential multiple thermocouples connected with a Leeds and Northrup potentiometer recorder. As Fig. 1 shows a continuous flow of the air to be analyzed is maintained by a motor-driven rotary pump through a suitable purifying train of concentrated sulfuric acid, charcoal,

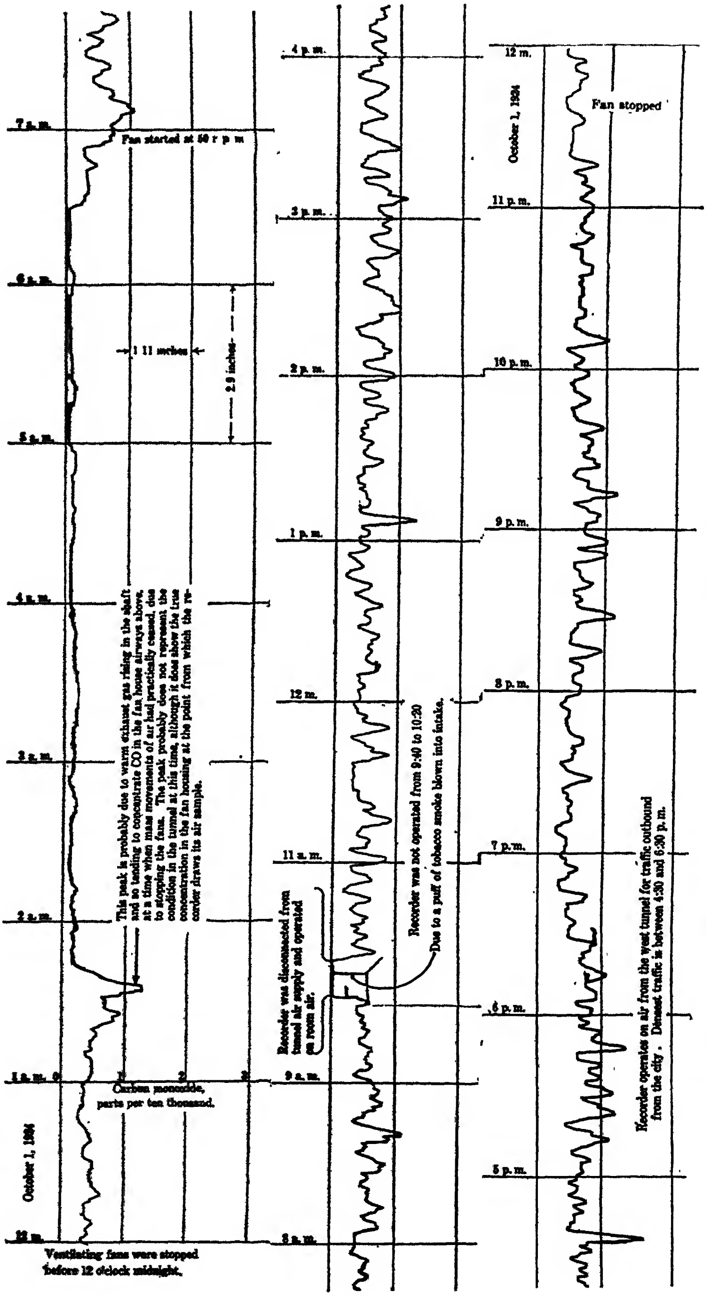

FIG. 3.—Typical record for one day of analyses for carbon monoxide in air of Liberty Tunnels, Pittsburgh, Pa.

soda-lime and calcium chloride; and then through the catalyst which is maintained at a constant temperature by a surrounding water bath. An automatic flowmeter keeps the air current constant. The catalytic material is hopcalite ($CuO + MnO_2$), which is the material used in carbon monoxide gas masks. Concentrations are expressed on the records in parts by volume of CO per 10,000 parts of air-gas mixtures, and the recorder is sensitive to 1 or 2 parts per million. Trial operations in the laboratory for 7 to 8 mo. analyzing air that contained exhaust gas from a gasoline engine have proved its reliability and economy.[5]

Fig. 2 shows an automatic carbon monoxide recorder installed in the fan house of the Liberty Tunnels[6] at Pittsburgh, Pa. The instrument is equipped with a bell to warn the attendants should the carbon monoxide at any time exceed 4 parts in 10,000 through fire, accident or excessive traffic in the tunnel. If desired the bell-ringing system can be designed to operate a relay switch to increase ventilation automatically by speeding up the fan motors. This recorder operated continuously and satisfactorily for 12 mo. Its use permitted the saving of considerable power and at the same time it safe-guarded the public by insuring adequate ventilation at all times.

Fig. 3 is a typical record of analyses for one day.

The records of the Liberty Tunnels usually showed less than 1 part carbon monoxide in 10,000 parts of air. With such records, the operating officials can safely combat any claims for damages to health due to inadequate ventilation that might be made by unscrupulous or misguided persons.

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[5] S. H. Katz, D. A. Reynolds and J. J. Bloomfield: A Carbon Monoxide Recorder and Alarm. *Tech. Paper* 355, Bureau of Mines (1926).

[6] A. C. Fieldner, S. H. Katz, and E. G. Meiter: Continuous CO Recorder in the Liberty Tunnels. *Eng. News-Record* **95** (1925), 423–424.

The Holland Tunnel (The Hudson River Vehicular Tunnel)*

By Ole Singstad,† New York, N. Y.

The legislatures of New York and New Jersey, determined in 1919 that a vehicular tunnel should be built under the Hudson River. On July 1, 1919, an engineering staff was organized with the late Clifford M. Holland as chief engineer. After preliminary investigation it was decided to build the tunnel from Canal St., New York, to 12th St., Jersey City, which is the center of the heaviest traffic across the Hudson River. It was also decided to build two tunnels—one for west-bound traffic and one for east-bound—the tunnels to be of cast iron, built up of segments and bolted together, each tunnel to have a roadway 20 ft. wide and a clear head room of 13 ft. 6 in. The tunnels have an exterior diameter of 29 ft. 6 in. The interior lining is of concrete, the sidewalls, tiled with a vitreous white tile, and the roadways are paved with granite block. The entrances and exits at both ends are separated by two blocks so as to reduce traffic congestion.

Basis of Ventilation

It was recognized very early that one of the most important problems confronting the engineers was a proper ventilation system. It was estimated that each tunnel as planned would have a traffic capacity of 1900 vehicles per hr. The tunnels are 8500 ft. long between portals, and it was considered that the ventilation should be based on all of these vehicles being operated by gasoline engines. That presented a problem in ventilation unlike any other either in magnitude or character.

There have been vehicular tunnels built in Europe, principally in London, but they are shorter and have a smaller traffic capacity. Up to the present time, at least, they have handled a very large percentage of horsedrawn vehicles so that the necessary ventilation has been provided for by the natural draft through the portals and open shafts. Within the last few years, however, a small mechanical ventilating plant has been installed in the Blackwall Tunnel, which has been in operation about 27 years.

I might mention that at the time this work was started, it was considered by many that the problem of ventilation was insuperable and it

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* Address at New York Meeting, February, 1926.

† Chief engineer, New York-New Jersey Tunnel Commissions.

was therefore suggested that vehicles should be taken through the tunnel by other than their own motive power, such as moving platforms. Serious consideration was given to these means but it was found that they all presented so many difficulties in their application and so materially reduced the traffic capacity of the tunnel that they were abandoned.

Division of Research

The ventilation problem was studied from the start under three main subdivisions: First, the amount and composition of exhaust gases given off by motor vehicles as ordinarily operated on the road; second, the physiological effects of these gases and, third, the method of ventilation and the power requirements.

A thorough search was made of all the available information bearing on the amount and composition of exhaust gases. The most poisonous constituent of the exhaust gas is carbon monoxide, and the presence of that is due to incomplete combustion. The degree of combustion depends upon a number of variable factors and this could be determined only experimentally. As there was no adequate information available that would serve as a guide for the solution of this problem it was necessary to perform extensive tests on motor vehicles.

With respect to the second phase of the problem, the physiological effect of the motor exhaust gases, we also made a search of the available information. Although it appeared certain that the dilution necessary could be set at a sufficiently low figure so that the tunnel could be used with absolute safety and comfort, still the question of the large quantities of air required to ventilate a tunnel under such heavy traffic made the power bill an important consideration. Therefore it was considered necessary to determine by extensive research and experimental work exactly what dilution was necessary to provide safety and comfort.

Difficulty in Handling Air Flow

On the basis of the preliminary information it was apparent that the quantity of air necessary to ventilate the tunnel adequately could not be handled in the ordinary way of ventilating a tunnel—that is, by blowing it in at one end, through the tunnel and out the other end. If that were done it would create such a gale of wind that it would be uncomfortable and even dangerous to use the tunnel. It was therefore decided that the ventilation could be accomplished by the supply of air through a duct under the roadway with withdrawal through a similar duct. Here again it was found that there was not available sufficiently reliable information to determine the friction losses in concrete ducts of such size and length, and it was deemed necessary to carry out independent experimental and research work.

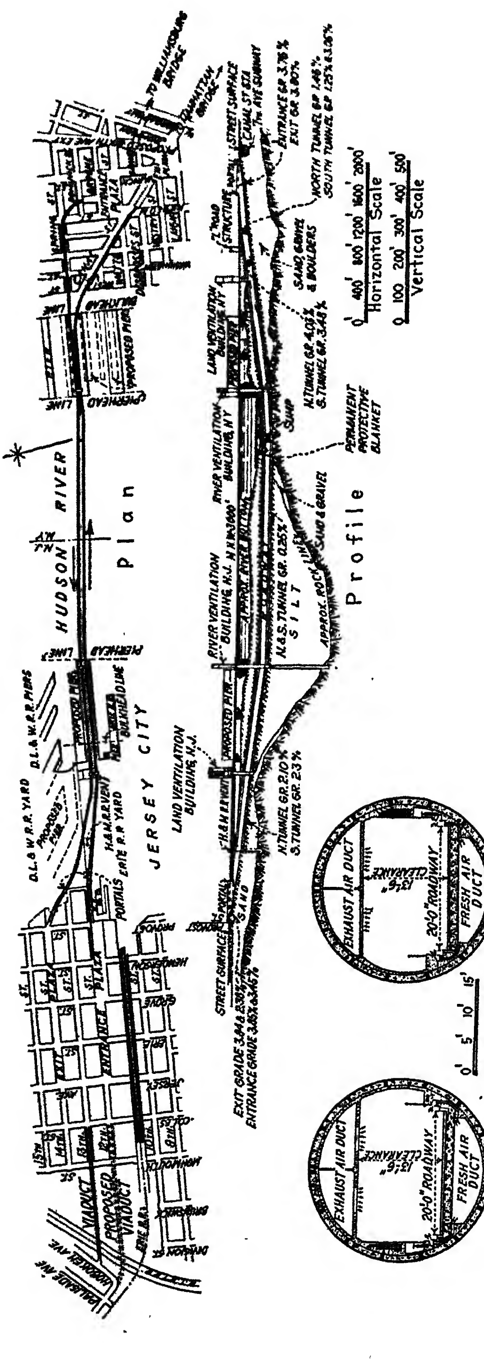

FIG. 1.—PLAN, PROFILE AND CROSS-SECTION OF THE HOLLAND TUNNEL.

This work, was done through a cooperative agreement between the New York-New Jersey Tunnel Commissions and the U. S. Bureau of Mines whereby the Bureau was to carry out the investigations as outlined. The Bureau's contribution of technical services, labor and supplies was substantial.

Composition of Exhaust Gases

The first test to determine the amount and composition of exhaust gases was performed by the Bureau at the experiment station in Pittsburgh, under the direct supervision of A. C. Fieldner. About 2000 tests were made on more than 100 different types and sizes of motor vehicles on grades corresponding to the grades of the tunnel—light and loaded, up and down grades at various speeds. By these tests the first question was satisfactorily answered. We learned definitely the amount and composition of exhaust gases from motor vehicles as operated under ordinary road conditions.

The series of tests to determine the physiological effects were performed for the Bureau at Yale University, under the direction of Dr. Yandell Henderson. As a result of these tests it was concluded that if carbon monoxide is kept down to 4 parts in 10,000 parts of air the air will be entirely satisfactory for an exposure of 1 hr.[1] As a car would go through the tunnel in about 10 min., that is an ample margin; and even for a truck, which would take 15 min., there would still be ample safety.

The third series of tests was performed by cooperative agreement, involving the New York-New Jersey Tunnel Commissions, the Bureau of Mines and the University of Illinois Experiment Station. A concrete tunnel was erected on the campus of the University, and a fan having a capacity of more than 100,000 cu. ft. per min. was installed. The friction losses in the concrete duct were established for the air flowing all the way through, the air being taken off or entering at intervals such as would be the case in the tunnel. This work was done under the direction of Prof. A. C. Willard.

Tests in Model Tunnel at the Experimental Mine

After the three questions have been satisfactorily answered, we felt certain that the problem of ventilation had been solved, but in order that there should be no question it was decided to demonstrate the practicability of this method of ventilation in a model tunnel. That work was done, as the previous work had been done, by the Bureau of Mines by a cooperative agreement.

[1] These results were later confirmed by extensive physiological tests carried on under Dr. R. R. Sayers, chief surgeon of the Bureau of Mines, in the course of the later testing in the model tunnel at the Experimental Mine.

A tunnel, of a cross-section about one-third that of the Holland Tunnel, was constructed in the Bureau's Experimental Mine at Bruceton, Pa., about 1100 ft. in the hillside and shut off from the outside air except by intake and return cross-cuts. Eight automobiles were driven on the elliptical roadway for a period of 1 hr. in each test, during which the air was constantly sampled in various ways, temperature and humidity readings taken, and physiological tests made on the 50 or more persons taking part in the tests.

Taking off the Exhaust Gases

There was one very important question which had not been definitely settled although there had been considerable discussion of it. Many of us believed that the proper procedure was to supply fresh air at the roadway and take the vitiated air off at the top, but many people thought that we should take the gases off at the roadway where they are expelled from the engine. This seemed rather plausible, but the tests at Bruceton very definitely established that better results were obtained by introducing the air at the bottom.

The Ventilation Stations and System

There are four ventilating stations. One in the Erie R.R. yard, Jersey City, one in the river near the New Jersey pierhead, one near the pier line, New York, and one at Washington St., New York.

There is traffic in one direction only in each tunnel. The air enters the tunnel roadway through continuous openings from a continuous chamber on either side of the roadway, which is supplied through openings from 10 to 15 ft. apart leading from the fresh-air duct located under the roadway. (Fig. 1.) The chamber has a steel plate in front of it leaving an opening varying in width from ¾ to 1½ in., depending upon whether the tunnel is on the downgrade,[2] level, or upgrade. The slots are arranged so a continuous stream of fresh air enters the tunnel on either side of the roadway through the entire length. This air mixes with the gases, which rise and pass through the openings of the exhaust ducts.

Fan Equipment

The fan equipment installed in each of the four ventilator buildings ventilates the tunnel halfway to the next building. The fresh air is taken in through louvered openings in the sides of the buildings and down the shafts to the duct under the roadway, where it is taken off at frequent intervals as already explained. The ducts are bulkheaded off halfway

[2] The tunnels make 4 per cent. on the downgrade and a maximum of 3.8 per cent. on the upgrade.

between ventilating buildings. The air is drawn into the exhaust duct by the negative pressure created by the exhaust fans, passes through the air-tight fan chambers in the buildings and is exhausted through stacks extending about 20 ft. above the roof of the buildings.

Carbon Monoxide Dilution

On the basis of the estimated number of vehicles that will use the tunnel when operated to capacity and in order to have a dilution whereby the carbon monoxide will not exceed 4 parts in 10,000, it is estimated that 3,761,000 cu. ft. per min. of fresh air will have to be introduced. This air expands somewhat due to the rise of temperature, and the quantity exhausted will be somewhat larger than that.

We are using the backward-curved blade type of fans as these must be operated in parallel, the installation being on the basis of three fans to each duct. These fans are operated by wound rotor a.c. motors. This will permit the necessary variation in the air supply. Two of the three fans on each duct are required at maximum traffic operation. The third fan is always standing by as a spare unit. Intermediate quantities of air can be obtained by cutting out one fan or running two fans at some intermediate speed.

Fan Capacity and Power Supply

The power for operating the ventilation equipment will be obtained from both sides of the river. There will be three independent cables from each company, connected with at least two independent generating sources on each side. That will give four independent generating sources and six independent cables, each cable having a capacity sufficient to carry the full tunnel load. The power from either side is carried through the tunnels, so that all fans can be operated from any source of power. The electric current is supplied at 13,200 volts and it is transformed down to 2300 for the larger fans and 440 for the smaller fans.

The installed horsepower for the fan equipment will be about 6000; about 4000 hp. or two-thirds of the installed capacity will be required to ventilate the tunnel at maximum capacity.

There are 84 fans in all, half of them blowers and half exhausters. Fifty-six will be required when the tunnel is operated to its maximum capacity and 28 will be standing by as spare units.

The capacity of the fans varies from 81,000 to 227,000 cu. ft. per min., and they operate against static pressures varying from ½ to 3¾ in. because of the different lengths of ducts. For instance, the fans ventilating the section of the tunnel from the river shaft halfway up to the land shaft have only 700 to 800 ft. of tunnel to ventilate, while the ones

at the opposite side of the river ventilating building have over 1700 ft. to ventilate. The greatest tip speed is about 9000 ft. per min.

Characteristics of Ventilation System

The characteristics of the ventilation plan that has been adopted can be briefly stated as follows: The fresh air is supplied throughout the entire tunnel so that the air will be just as pure under the middle of the river as it will be near the portals.

The air supply at any point can be regulated to meet requirements. In other words, it is possible to provide necessary ventilation for the requirements on the downgrades, on the upgrades and on the level. There is no discomfort due to high-velocity air currents. The tunnel atmosphere has a complete change every 1½ min.; that means 40 complete changes per hr. The air stream comes out through the slits so gently that if you hold your hand about 5 ft. away you cannot feel the movement. Driving through the tunnel one will not feel even a slight breeze.

The ventilation of the tunnel will not be affected by outside wind conditions. If air were blown through there might be a head-on wind at the exit. This will not hold true with the method employed. The exhaust gases are immediately caught by the fresh air stream and diluted so there will be immediate diffusion.

Safety from Fire

Personally, I am of the opinion that the tunnel could not have been successfully ventilated by any other method than that described. It is also the only really safe method from a fire hazard point of view. If there was a swift air current blowing through the tunnel and a fire broke out—as is certain to occur because vehicles going through the tunnels will not be immune from fire any more than they are in the streets—a very serious situation might result. The flames might spread from one vehicle to the next and would also carry the smoke through the tunnel so that conditions would be unsafe and very apt to create panic. The method used permits no longitudinal air currents. In the demonstrations with smoke bombs at the experimental tunnel it was found that although the smoke was so dense the men could not see their hands in front of their faces it did not spread more than 30 ft. on either side of the source of the smoke, and we expect that in the tunnel we will have a similar condition. The traffic in front of the car on fire will naturally keep on moving. Behind the fire there will be sufficient room for traffic to back up. It will not be necessary to go more than about 30 ft. to be outside the zone of the smoke.

Compressed Air Tunneling

The tunneling operations for the greater part were done under compressed air. The tunnel was started on the New York side from the first

ventilating station, which is at Washington St. The shafts were first sunk as pneumatic caissons and the bottoms sealed. The shields were erected in the lower part of the caissons and temporary roofs placed in the latter and the chambers put under compressed air. Openings were provided in the caissons before sinking, the temporary steel bulkheads closing these openings were removed and the shields started on their way under the river.

Construction and Sinking of River Caisson

The steel boxes for the river caissons were built on shipways up to an elevation just above the drums. Each caisson has an air-tight and water-tight steel roof which during sinking served as a working chamber for the men. They were then launched and towed up the river and located above their sites. They were then gradually built up and concreted so that they settled down to the river bottom and the excavation carried on under compressed air.

The sinking of these caissons was accomplished while the shields were being driven from the land shafts. By the time the shield approached the shaft the caisson had come down to its final position, the concrete bottom placed and the steel roof moved to its upper position and the entire chamber placed under compressed air. When the shield approached the caisson the temporary bulkhead was burnt out and the shield pushed into the shaft. The corresponding bulkhead on the opposite side was then removed permitting the shield to continue.

Solution of Foundation Problems

We had a very difficult foundation problem at the New Jersey river shaft. The caisson is 107 ft. below water level and is in soft silt. The silt could not sustain the load and it was necessary to carry the foundations to rock 260 ft. below the water level. This was accomplished by sinking eighty-four 24-in. diam. steel shell-reinforced concrete piles. There are two shafts on the New Jersey side at this point because of the peculiar conditions governing the design of the pier. On both shores, the tunnels and the ventilation shafts are protected by new piers. These were constructed on the New York side by the New York City Dock Dept. and on the New Jersey side by the Erie Railroad Co.

The piles were about 155 ft. long below the cutoff. They were sunk from the platform above the water surface by using a churn drill and chopping a hole ahead of the pipe. The pipe came in 20-ft. sections, and each section was put down in the hole that was churned off. Then more hole was churned and another section put on. Finally rock was reached. The hole was cleaned out carefully and an 8-ft. concrete plug placed under water. After the concrete plug had set the pipe was unwatered,

the steel reinforcement placed inside of the pipe and the concrete deposited up to the elevation of the subgrade of the shaft. At that point the pipe was sawed off by dropping circular cutters and cutting off the pipe at the proper elevation. When all of the pipes were down and sawed off the caisson was sunk to the top of the piles.

Structure of Tunnel

The tunnel is built of cast iron rings and was erected under the protection of a shield. The latter is a steel cylinder with an internal diameter in the tail slightly larger than the outside diameter of the tunnel, which leaves about 3 in. from the completed tunnel iron and the inside plates of the tail of the shield. The tail is long enough to cover 2½ completed rings. Each ring is 30 in. long. To the rear is a concrete bulkhead 10 ft. thick in which were the locks. In the lower part of the bulkhead are two locks for material and in the upper two locks for men. One was used for letting the men in and out and the other was for emergency with the door facing the heading always open. This section of the tunnel was always under compressed air of sufficient pressure to hold the water out of the heading. The operation consisted of driving this shield ahead by hydraulic jacks, each jack having a capacity of 200 tons reacting against the completed part of the tunnel. The tunnel segments weigh 1½ tons each and were put in place by a hydraulic erector.

Classes of Materials Penetrated

The tunnel passes through various classes of material. On the New York side there was sand to start with and while tunneling through this the face of the heading was protected by breast boards. After the completion of the shove and during the installation of the iron work, the miners would remove the top board carefully and scoop the material into the tunnel, set the top board ahead 30 in. and so on with the next board. The breast boards were supported on the shields by hand jacks. The tunnel passed through rock for a distance of about 800 ft. near the New York pierhead line.

Working through Silt

In silt the operation was different. In going through sand all of the material that is displaced has to be taken in through the tunnel heading and disposed of through the tunnel. In the Hudson River silt, which is so soft that it can be pushed aside, the shield was closed with steel doors. The steel framing in the shield was horizontal and vertical to give the necessary stiffness, and each compartment was closed with a door. In going through a large body of silt in the river the doors were closed

and the shield forced ahead, the material being displaced, except for one or two pockets, where the door had to be opened in order to guide the shield. Such a tremendous pressure was set up in the ground that it became almost impossible to guide the shield by varying the pressure on the jacks and it was guided by opening the doors. There was a safety runway at the elevation of the springing line, so in the case of flooding the tunnel, the men could run back to safety.

Blowouts and Their Cause

Only one bad blowout occurred. Then the tunnel was flooded but there was no loss of life. The reason for the blowout was that in going through the rock reef on the New York side it was necessary to blast out the rock. The top of the tunnel was in soft ground and on account of the unbalanced pressure at the top of the tunnel when there was enough air pressure to keep the water out at the bottom there was a great tendency for the air to blow. There was a constant blowing out of air and this air escaping weakened the river bed until it finally blew. The men had sufficient warning and all escaped into the emergency lock and closed the door in time. After four days the tunnel was pumped out and tunneling operations resumed.

During construction there were provided at intervals in the compressed air chamber safety screens extending down from the top to about the horizontal diameter so in case the tunnel was flooded there would be an air pocket in which the men would not drown.

Tunnel Unusually Dry

An alloy steel bolt was used on this tunnel for the first time in tunnel construction and it permitted the use of a very much smaller bolt than would have been required if ordinary carbon steel had been employed.. It also made it possible to get the tunnel tight. The fact that there is less deformation in this tunnel than in tunnels of smaller size is due to the rigid bolting requirements, which were made possible by the high quality steel used in the bolts.

While tunneling in silt the upper part of the shield was closed at first and the silt permitted to flow in at the bottom. This method caused some difficulty and the tunnel rose during the driving due to the excessive pressure set up. Consequently a different method was devised for controlling the shield. In order to overcome the tendency to rise it was decided to leave the material taken in the tunnel until the disturbance caused by the driving had subsided. That gave greater weight to the tunnel and also saved expense. A bulkhead was erected which trailed along with the shield over which the material flowed, down to the lower

part of the tunnel cross-section. It was later excavated and brought out under normal air pressure.

Chief Engineers of the Holland Tunnel

One man, more than anyone else, is responsible for having brought the project to the state where it is today and that is the first chief engineer, Clifford M. Holland, who died Oct. 27, 1924, two days before the New York and the New Jersey headings were scheduled to meet under the river. The Commissions, recognizing Mr. Holland's work and sacrifices—no doubt he gave his life to the job—named the tunnel The Holland Tunnel.

Mr. Holland was succeeded as chief engineer by the engineer of construction, M. H. Freeman. He also died, less than five months after Mr. Holland.

DISCUSSION

George S. Rice,* Washington D. C.—This project was such a great engineering undertaking it had to be considered most carefully. Since The Holland Tunnel was decided upon in 1919, various vehicular tunnels have been started and the Liberty Tunnels in the South Hills of Pittsburgh have now been completed. While these were about 6600 ft. long, they were comparatively simple as an engineering undertaking; there were no limitations of size and the grade was uniform. These tunnels use a system of ventilation that would not have been applicable to The Holland Tunnel.

The vehicular tunnel between Oakland and Alameda, Cal., is under construction. Its first consulting engineer was, also, Clifford M. Holland, under whose advice the same general design of construction as The Holland Tunnel was selected. A combination bridge and tunnel under the southern arm of San Francisco Bay, of similar design, has been recommended by consulting engineers but the project has not yet matured.

The writer was fortunate in being able to arrange, with the approval of the Director of the Bureau of Mines, for several phases of the cooperation on the fundamental factors affecting the control of the design of the tunnel, and was particularly concerned in the method of transverse rather than longitudinal ventilation and especially in the removal of the exhaust gases from the roadway as rapidly as possible. He also was called on by Mr. Holland in planning control of ventilation in case of fires

* Chief mining engineer, U. S. Bureau of Mines.

following collisions. Such collisions and fires are liable to occur and unless there is control of ventilation there might be danger to life from extensive explosions, and from the smoke of fires which can now be drawn off quickly into the return compartment.

This method of ventilation is well worthy of consideration in special cases of mining. The methods of driving the tunnel, to which Mr. Singstad refers, are also of great interest to mining engineers.

A. C. Fieldner,* Pittsburgh, Pa. (written discussion).—One of the problems in designing the ventilation for The Holland Tunnel was the determination of the quantity of carbon monoxide (the asphyxiating constituent of exhaust gas) given off by automobiles and trucks under tunnel conditions of speeds and grades. Two methods of solving this problem were considered, viz.: (1) The construction of a model tunnel with treadmill bottom in which cars could be operated and the air contamination determined by analysis at various known rates of air change, and (2) road tests of automobiles in which the amount and composition of the exhaust gas is determined by analyzing average samples of the undiluted exhaust gas taken from the engine manifold, and calculating the quantity from the measured amount of gasoline consumed.

Carbon Monoxide Tests in Liberty Tunnels

The second, or road test method, was selected as the most practical and representative method, since the actual road conditions as to grade, speed, carburetor adjustment, etc., can be reproduced, and the results in terms of quantity of carbon monoxide evolved give the actual figures needed in the ventilation computations. However, with the completion of the Liberty Tunnels in Pittsburgh, an opportunity was afforded to check the results of the second, or road test, method by method one—*i. e.*, Liberty Tunnels with the ends boarded up, could be used as huge gas-tight test chambers nearly 6000 ft. long, in which motor cars and trucks could be operated at any desired speed and the atmosphere analyzed for carbon monoxide to see if the degree of contamination found was in agreement with the values calculated from the previous road test data.

These tests[3] were made in January, 1924, with the aid of A. D. Neeld, engineer for Allegheny County, in charge of construction of the tunnels. The twin parallel tunnels (Fig. 2) are 5888 ft. long; each has a cross-sectional area of 468 sq. ft. and a total volume of 2,755,584 cu. ft. They are connected at evenly-spaced distances by 11 man passageways, which are closed at each end by a steel door. At approximately mid-point of each

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* Chief chemist, U. S. Bureau of Mines, and superintendent, Pittsburgh Experiment Station, Pittsburgh, Pa.

[3] A. C. Fieldner, W. P. Yant, and L. L. Satler, Jr.: Carbon Monoxide under Traffic in Liberty Tunnels, *Eng. News-Record* (1924), **93**, 1222.

tunnel are two ventilating shafts, the upcast opening directly into the roof, and the downcast into a nozzle parallel to the upper periphery of each side; the distance between the nozzle and the shaft opening is approximately 75 ft. At the time of these tests the shafts were tightly closed with boards covered with roofing paper. The only other openings into each of the tunnels are the end portals, which were also closed at this time with a tightly fitting wooden bulkhead having at the bottom and near the center of the roadway an opening 9 by 12 ft. for admitting the automobiles.

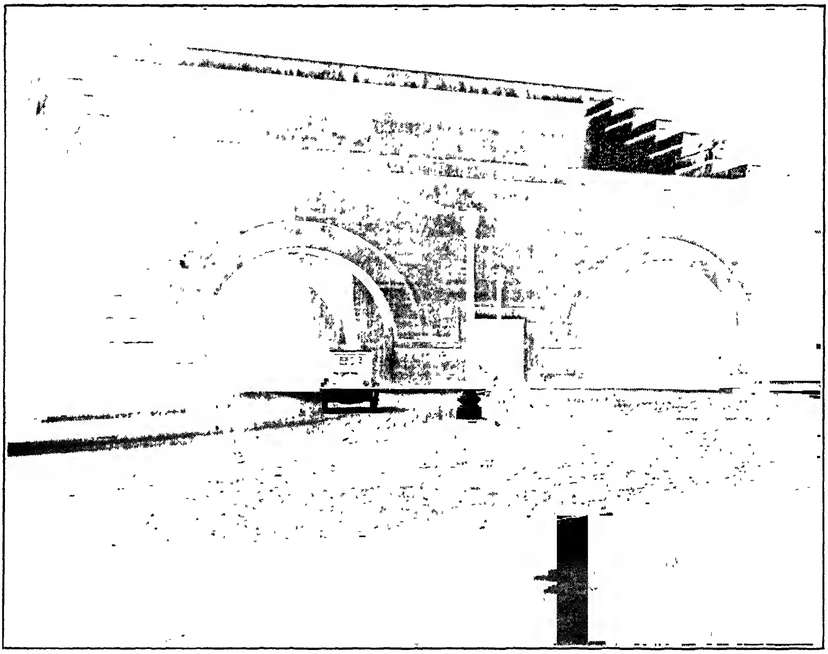

FIG. 2.—PORTAL OF THE LIBERTY TUNNELS, PITTSBURGH, PA.

In general, the tunnels as arranged for the tests constituted two huge gas chambers, which the machines entered through one of the bulkhead openings and left through the other. These openings were not closed after each succeeding machine, but were closed at the end of each test, previous to sampling the atmosphere within.

PROCEDURE OF MAKING TESTS

The general procedure of making the tests was to allow a predetermined number of cars to pass through the tunnel, after which the doors were closed and the samples of tunnel atmosphere quickly obtained. The total number of cars permitted for each test was that number which, on the basis of road test data, would produce an atmosphere of approx-

TABLE 1.—*Carbon Monoxide in Tunnel Air and in Blood of Observers*

Direction of Cars from Station 6 to Station 1 in East Tunnel; from Station 1 to Station 6 in West Tunnel

Tunnel	Air Analysis in Tunnel Air						Blood Analysis and Symptoms				
	Stations						Number of Men	Average Per Cent. Saturated with CO	Duration of Exposure		Symptoms
	I	II	III	IV	V	VI			Hr.	Min.	
Test 1							3	17	1	20	No symptoms. Canaries at Station 1 and 2 showed no symptoms.
East	0.041	No sample	0.018	0.014	No sample	0.013					
West	0.016		0.020	0.028		0.043					
Test 2							7	21	1	20	Two men reported slight symptoms. Canaries O. K.
East	0.076	0.050	0.047	0.033	0.023	0.022					
West	0.017	0.021	0.033	0.055	0.066	0.077					
Test 3							8	18	2	30	No symptoms. Canaries O. K.
East	0.029	0.025	0.020	0.019	0.012	0.008					
West	0.011	0.016	0.022	0.025	0.030	0.025					
Test 4							8	22	1	55	Two men reported slight, but not distinct symptoms.
East	0.039	0.043	0.027	0.021	0.013	0.012					
West	0.011	0.016	0.028	0.042	0.047	0.048					
Test 5							7	14	3	0	No symptoms.
East	0.033	0.023	0.018	0.018	0.014	0.005					
West	0.018	0.031	0.020	0.012	0.005	0.005					

Station 1— 567 ft. from north end of tunnel.
2—1567 ft. from north end of tunnel.
3—2567 ft. from north end and approximately 300 ft. north of shaft.
4—2222 ft. from south end and south of shaft.
5—1222 ft. from south end of tunnel.
6— 472 ft. from south end of tunnel.

imately 4 parts of CO in 10,000 of air, the maximum safe concentration of CO for an exposure of one hour.

In Test 1 (passenger cars) and Test 2 (trucks), the cars were solicited from public-spirited citizens, and, prior to the start of the test, were assembled in procession outside the portals, so that after the test had started a continuous and fairly evenly-spaced traffic could be maintained. In the remaining three tests (3, 4, and 5) this prearrangement was not made, the tunnels being merely opened for public use, the traffic depending on what happened to come along. In all cases an accurate record was made of the number and type of car. In addition to this, air velocity data taken with anemometers were recorded throughout the test, and temperatures and humidity readings were taken.

The gas samples were taken at the stations and positions shown in Table 1. The blood of all observers who remained in the tunnel for the duration of the test was examined for CO content. The results of these analyses are given in Table 1.

Calculated Concentration.—The CO concentration was calculated from the average values obtained in the 1920–21 road tests, which are given in Table 2.

TABLE 2.—*Carbon Monoxide per Hr., Cu. Ft., at 65° F. and 29.92 In. HG., for Vehicles Loaded to Half Capacity*[a]

Class of Car	Speed, Miles per Hr.					
	6	10	15	20	25	30
1. 5-passenger	...	46	61	73	[b](85)	(95)
2. 7-passenger	...	71	105	137	(155)	(168)
3. Trucks up to 1½ tons	...	60	72	94	(110)	
4. Trucks, 1½ to 3 tons	68	104	104	(130)		
5. Trucks, 3½ to 4 tons	93	148	131	(160)		
6. Trucks, 5 tons and over	110	152	(195)			

[a] A. C. Fieldner, A. A. Straub, and G. W. Jones: Ventilation of Vehicular Tunnels, *Jnl.* A. S. H. & V. E. **32** (1926), 17.

[b] Figures in parenthesis ( ) are extrapolated values.

The calculated concentrations in the tunnel for the number of vehicles in the tunnel tests are given in Table 3, together with the actual concentrations obtained by analysis of the tunnel atmosphere.

The latter were in all cases taken as the highest found for a single cross-section. Although it was desired to make the tests as nearly as possible without ventilation and loss of gas, the movement of the cars set up considerable air motion in the direction the cars were going, except in Test 5, where in the west tunnel the natural wind velocity opposing traffic was greater than that resulting from the cars. However, in this

TABLE 3.—*Comparison of Calculated and Actual Carbon Monoxide Concentrations in Liberty Tunnels*

Tunnel	Duration of Test Period (Min.)	Approx. Speed, (M.P.H.)	Number and Type of Cars: 5-Passenger Cars	7-Passenger Cars	Trucks up to 1½ Tons	Trucks 1½–3 Tons	Trucks 3½–4 Tons	Trucks 5 Tons and Over	Total Number of Cars	Total Volume of CO Liberated Cu. Ft.	CO in Tunnel Air by Calculation	CO in Tunnel Air by Analysis
				Test 1, Jan. 12, 1924								
East.........	60	25	111	86	4	1	...	1	203	1046	0.038	0.041
West.........	60	25	114	85	...	...	...	...	199	1010	0.037	0.043
				Test 2, Jan. 17, 1924								
East.........	56	15	4	2	13	40	4	52	115	1242	0.046	0.076
West.........	56	15	5	3	12	...	36	56	112	1289	0.048	0.077
				Test 3, Jan. 21, 1924								
East.........	135	30	69	36	38	7	2	...	152	724	0.027	0.029
West.........	135	30	53	54	47	6	...	6	166	888	0.032	0.033
				Test 4, Jan. 24, 1924								
East.........	120	30	151	72	40	20	6	10	299	1539	0.056	0.043
West.........	120	30	173	73	26	5	...	13	290	1422	0.052	0.048
				Test 5, Jan. 28, 1924								
East.........	180	30	271	89	27	4	6	7	404	1824	0.067	0.033
West.........	420	30	212	72	75	15	1	23	398	2118	0.078	0.031

test the traffic per unit of time was low. It is quite probable that an increased number of cars would have set up a movement in the direction of traffic, this having occurred in all previous tests. Just how much of the gas was lost through the openings in the bulkheads is difficult to estimate, because measurements of air velocity indicated not a progressive unidirectional movement with the cars, but often a reversal, indicating a back-slipping of the air previously set in motion resisted by the bulkhead. However, the net result was in the direction of traffic, which is further substantiated by the gradual variation in the average results of analysis (Table 3) obtained for each cross-section. This gradation is indicative of forward air motion and dilution with air from the entering portal. Accordingly, the section having the highest concentration would be the least contaminated, and the concentration value for the entire tunnel, if no dilution or ventilation had occurred, must have been at least this great.

Discussion of Results

With the exception of Tests 3 and 4, the agreement between the results obtained by analysis and those calculated is seemingly very good.

In Test 2, the amount found by analysis far exceeds that calculated. The large 5-ton trucks used were loaned by contractors, and the drivers, for whom these tests were a sort of gala occasion, continually drove in rhythmic acceleration and at an unsteady speed, which would result in numerous backfires and a much greater consumption of gasoline and liberation of CO. Also, the trucks were adjusted for heavy winter hauling, and quite likely the gasoline-air mixture was very rich.

Test 5 is not comparable with the others as too much gas was lost from the east tunnel through high wind velocity in the direction of the cars, and also the duration of the test was much longer than the previous tests. In all probability the time was a great deal longer than that required for a cross-section of maximum concentration to travel the length of the tunnel. The conditions for comparison in the west tunnel were even worse, as for the first and only time during these the air velocity due to high winds exceeded at times that produced by the cars, and the section of maximum concentration was repeatedly shifted. This reversal of direction, being changeable, allowed for incoming air to alternate between the ends of the tunnel, consequently markedly lowering the concentration of CO. Also, the duration of test in this tunnel was 7 hr., which allowed too much time for the escape of gas.

In all tests, the blood of the observers showed the presence of CO in amounts in agreement with what would be expected from exposure to CO in concentrations such as were found by the air analyses.

Taken as a whole the results of these tests show that the road test data on carbon monoxide are correct and may be regarded as fundamental data for calculating the ventilation requirements of vehicular tunnels.

A Member.—What provisions have been made for keeping the contaminated air discharged at one of those fan houses from circulating back to the inlet of the fan pumping supposedly fresh air down into the tunnel?

O. Singstad.—The exhaust stacks are carried up to about 20 ft. above the roofs of the buildings and the air shot out of the stacks at a considerable velocity, going up into the atmosphere much the same as smoke from a chimney. The fresh air is taken in through openings on the lower floors of the buildings and we feel certain that none of the vitiated air will return in that way.

G. W. Farny, Morris Plains, N. J.—What provisions have been made to take care of the humidity in the air. The amount of air that is sent to the tunnel is of such magnitude that naturally it carries a certain amount of humidity.

O. Singstad.—No special provision has been made for taking care of the humidity. The fresh air is to be taken into the tunnel in large quantities and at outside temperatures, but the relative humidity in the tunnel will be less than outside because there will be a slight increase in the temperature of the atmosphere of the tunnel. I do not believe there will be any tendency to condensation on the tunnel walls.

G. W. Farny.—I am still of the opinion that with the heavy fogs we have, as is the case in London, there will be a certain deposit of humidity on the walls of the tunnel.

O. Singstad.—We have made no special provision for eliminating the fog from the tunnel other than that there may be a tendency for the air to clear on account of the reduced relative humidity.

The agreement entered into between the Tunnel Commissions and the U. S. Bureau of Mines was a most happy solution of our problem. We have felt free to call upon the Bureau as often as new angles of the problem developed and had its heartiest cooperation. We owe a great deal to the splendid work done by the Bureau of Mines in the correct solution of the ventilation problem of this tunnel and we are very much indebted to it.

Occurrence of Fire Damp in Bituminous Coal Mines

By Frank Haas,* Fairmont, W. Va.

(Pittsburgh Meeting, October, 1926)

Many articles on the physical properties of fire damp have appeared in the Transactions and elsewhere but practically nothing has been written in regard to its occurrence or fluctuation in quantity in an ordinary mine. After the Monongah explosion in 1907, with the loss of 361 lives, the Consolidation Coal Co. sought a method by which such catastrophes could be prevented. While the cause of this particular explosion was attributed to coal dust, some of those acquainted with the mine thought that fire damp was involved, if it was not the initial cause. So the literature on the subject was studied and mines where similar conditions prevailed in this country and Europe, were visited, with the view of establishing some system by which gas could be more easily detected and controlled. The results of this study were disappointing for no method was found other than the time-honored fire boss with his safety lamp, with some variations in the rules and regulations governing his inspections and reports.

Duties of the Fire Boss

Pending the development of a better method, it was decided to standardize and perfect the duties of the fire boss and the reports based on his observations. Those who were then in service were closely examined, various lamps were experimented with and the lamps and individuals tested under known conditions. It was found that a fire boss could tell whether gas was present but his estimate of quantity was not even an approximation; therefore, while the services of fire bosses were indispensible there was need of additional information by which the quantity of gas could be estimated with some precision in order to determine its variation from time to time.

Chemical Process for Determination of Methane

After considering all methods known at that time a chemical process was adopted. The principles involved in the process are that in an enclosed receptacle, in the presence of excess air, methane will be completely burned by means of a hot platinum wire to carbon dioxide and

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water, and barium hydrate will absorb completely carbon dioxide in a definite relation.

The samples of mine air are taken by water displacement in copper cylinders 12 in. in length by 4 in. in diameter with conical ends and a stop cock at each end. In taking the samples, the can which is full of water, is held at the point where the air is to be tested and both stop cocks are opened. When the water has been completely drained, the stop cocks are closed and the can taken to the laboratory. In the laboratory, a definite volume (350 c.c.) is transferred to an Erlenmeyer flask. When the flask is filled with mine air a two-holed stopper, with the holes filled by glass plugs, is fitted into the neck of the flask. Ten c.c. of barium hydrate solution, of known strength, colored with phenolphthalein, is introduced into the flask by removal of one of the glass plugs and the insertion of a pipette. Shaking the flask periodically for 10 min. the excess barium hydrate is titrated with a standard solution of oxalic acid. The result is a factor of the carbon dioxide in the sample.

A second flask of the same size is filled from the same sample. The stopper closing this flask is fitted with two copper wires with platinum terminals and these are connected with a small platinum wire (30 gage). A current of about 5½ amp. heats the wire to a bright red, which causes the methane to burn to carbon dioxide and water. To prevent trouble due to expansion, the flask stands under water during the combustion. Combustion is complete in about 10 min. This sample is now treated with barium hydrate (see preceding paragraph; treatment same as for first sample) and the excess titrated with oxalic acid. The result of this test is a factor of the sum of the carbon dioxide originally in the mine air sample plus that due to the methane present. The difference is the factor representing the methane. The solutions are made up so that 1 c.c. is equivalent to 0.1 per cent. of carbon dioxide or methane. The oxalic acid solution is very stable. The barium hydrate solution is standardized each day.

Selection of Gas Inspectors

To carry out the work certain individuals are selected who have a general knowledge of coal mining. After they are thoroughly drilled in the principles and practice of ventilation they are given a course in the laboratory. They are then known as gas inspectors and are given certain mines or group of mines for active duty. These inspectors are frequently interchanged, partly as a check on the work, but more particularly to have the entire force familiar with all the mines. These gas inspectors are also first-aid training men, thoroughly familiar with, and capable of working with helmets and are subject to first call in case of explosion, mine fire or other accident.

The frequency of taking samples is limited to the working force available. After the first survey, mines that showed only traces of gas were sampled monthly; those containing perceptible quantities were sampled weekly; and those that showed the greatest quantities of gas were sampled daily.

The gas inspector when starting on his daily routine, carries an anemometer, a hygrometer and sufficient sample cans for the day's work. Eight cans are about all he can conveniently handle. His station for measuring velocity is constant for a long period and is of known cross-section. He takes 1-min. readings with his anemometer while moving uniformly across the section. A sample of mine air is also taken at this point, which is usually the end of a split. No difficulty is encountered in getting an average sample as it appears that the gas is thoroughly mixed with the air by the time the air current reaches the end of the split. There are, of course, frequent occasions for taking special samples such as determining the elevation of the explosive gas in the gobs or at intervals within a splint, but these are limited to the time available after the routine work is done.

Utility of Frequent Methane Determinations

The object of frequent determinations of methane is primarily to determine how much gas is evolved and to establish the relative volumes in the various sections with the view of supplying a sufficient total amount of air and distributing it to the various splits in a more logical manner. Mines having a small amount of gas do not need such close supervision but there are some coal mines where the production of coal is actually limited by the ventilating capacity, and it is in such cases that a most careful watch must be kept on this dangerous element with its erratic and apparently inscrutable tendencies. There was some hope that after daily analyses had extended over considerable time, an interpretation of the results might resolve itself into laws which govern its flow and possibly divulge some method by which the quantity could be predicted. After nearly 20 years of careful study in many mines the problem has not been solved, although certain tendencies have been established.

It would be a great assistance indeed, if a mining engineer were able to predict the quantity of gas that would be encountered in opening up a new mine. It would further help him if he knew in mines, already opened, when and under what circumstances additional gas would be encountered, or how to control the flow. A general statement can be made that the flow of gas follows the rate of development. Barometric pressure may have some influence on the flow of gas but, in the presence of other and more powerful factors, its effect is negligible. The truth is hidden in the confusion of many factors, most of which are unknown, and

we will have to content ourselves to deal with this troublesome element when and where and in the quantities in which it is found. The engineer is therefore forced to provide for maximum quantities and there still remains the uncomfortable thought that the best he can do may not be good enough.

Accuracy of Analysis

So far only daily samples have been taken and it is probable that a continuous record would throw additional light on the subject, particularly as regards daily fluctuations that may occur during active or idle periods. The cost of taking more than daily samples would be prohibitive; the accuracy of single determinations is most satisfactory. Frequent trials show that the same or different analysts can check within 0.005 per cent. The analysis as reported shows also the carbon dioxide but as the sample is taken by water displacement no dependence is placed on it. It does indicate however if any abnormal amount of the gas is present. In the presence of hydrocarbons, other than methane, these are necessarily reported in their equivalent of methane.

Source of Gas

The source of gas in coal mines has not been satisfactorily explained. Gases of occlusion, adsorption or even the volatile matter in the coal itself are insufficient to explain the enormous and continuous volumes of gas that are exhausted from the mines. Several years ago, in a paper before the West Virginia Coal Mining Institute, the theory was advanced that most of the hydrocarbon gases met with in the mines were natural gas and that coal seams were simply the reservoir and the path of flow from other strata. The suggestion was not generally accepted but our experience in the accumulation of quantitative results has strengthened this opinion.

The theory of coal formation assumes that all coal was originally in the form of peat—a compound of carbon, hydrogen and oxygen with some impurities. Like the peat of the present day, it had a high percentage of volatile matter of which a large part was water. In the transition to the various high-volatile and then to the low-volatile coals, it has lost the volatile portion in various degrees. It is assumed that this change is the result of time, pressure and heat. Time and pressure are indisputable as primary causes, but heat must be defined with the degree of temperature. For instance, there was no such action as takes place in a laboratory in determining the volatile matter. It is more probable that no destructive distillation took place at all. A close study of the ultimate analyses of coals will show that the transition from high-volatile to low-volatile coal can take place by successive removals of combined water with no loss of carbon whatever. Certainly no large loss, if any, of vola-

tile hydrocarbons was necessary to make the change. The physical characteristics were the result of time and pressure. Therefore gas as found in coal mines does not necessarily depend on coal as a source.

Graphic Presentation of Results

In the accompanying graphs are shown some of the results obtained. Individual mines were selected to illustrate particularly some of the

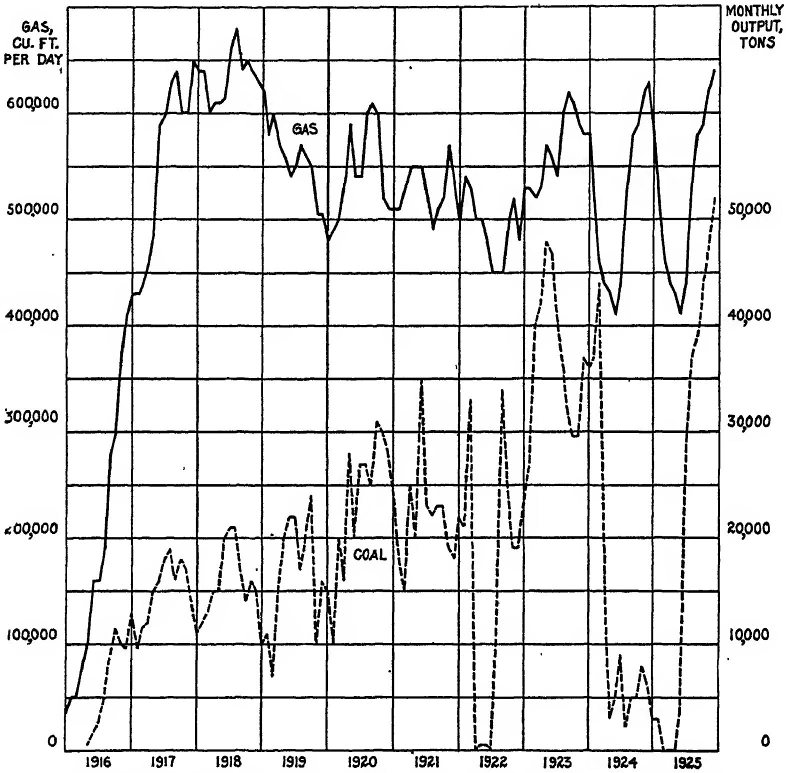

Fig. 1.—Relation of coal mined and volume of gas in a mine in Pittsburgh Seam, Fairmont Region, West Virginia.

factors which were supposed to predominate in the rate of flow. The extraction of coal must necessarily be considered and the rate of coal output has in every case been plotted in parallel with the gas flow.

Fig. 1 illustrates the gas flow (expressed in daily average by months for a period of 10 years) in a shaft mine 550 ft. deep in the Pittsburgh seam of coal in the Fairmont region of West Virginia. The coal is of the bituminous type with about 38 per cent. volatile matter. The seam is practically flat and is free from any irregularity, such as faults,

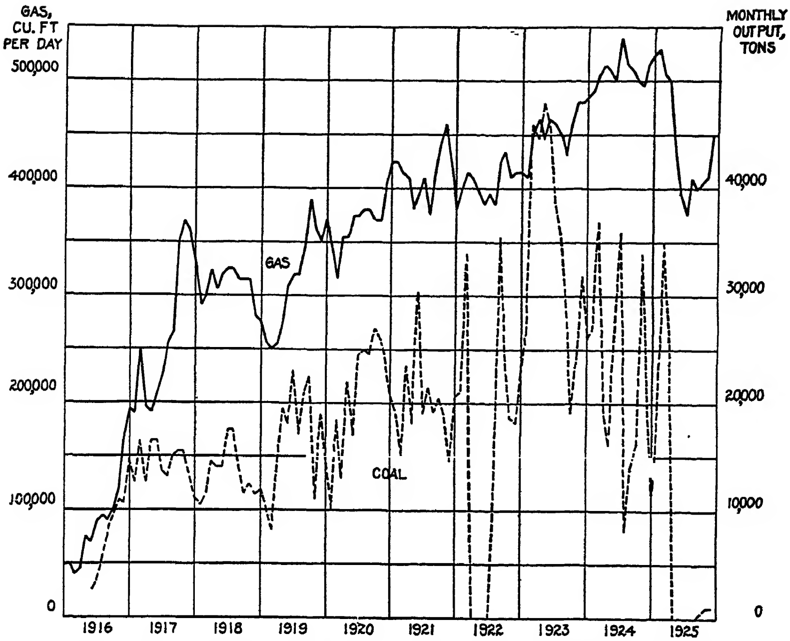

FIG. 2.—RELATION OF COAL MINED AND VOLUME OF GAS IN A MINE ADJOINING THAT OF FIG. 1. (SAME SEAM.)

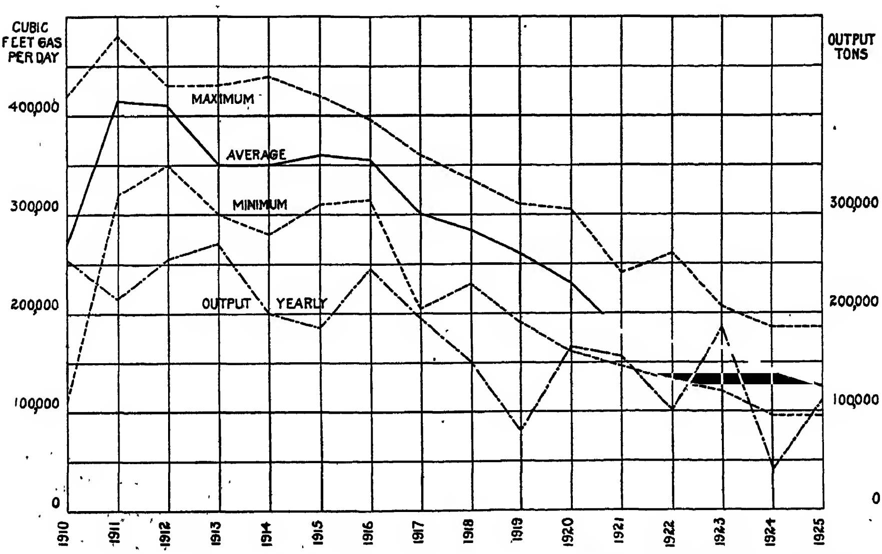

FIG. 3.—AVERAGE DAILY VOLUME OF METHANE IN A MINE IN PITTSBURGH SEAM, FAIRMONT REGION, WEST VIRGINIA.

or clay veins. The territory is practically virgin as the nearest opening into the seam was at least a mile away when the shaft was put down. The record has been continuous, samples taken daily (working days) since the shaft reached coal in January, 1916. The peculiarity of this mine is that the maximum flow of gas was encountered within 3 years of its opening when the flow settled down to a more or less uniform rate. In this case there is evidently no relation between the rate of output of coal and the flow of gas. Even in 1925 when there was no output for several months the flow of gas was not more than appreciably affected.

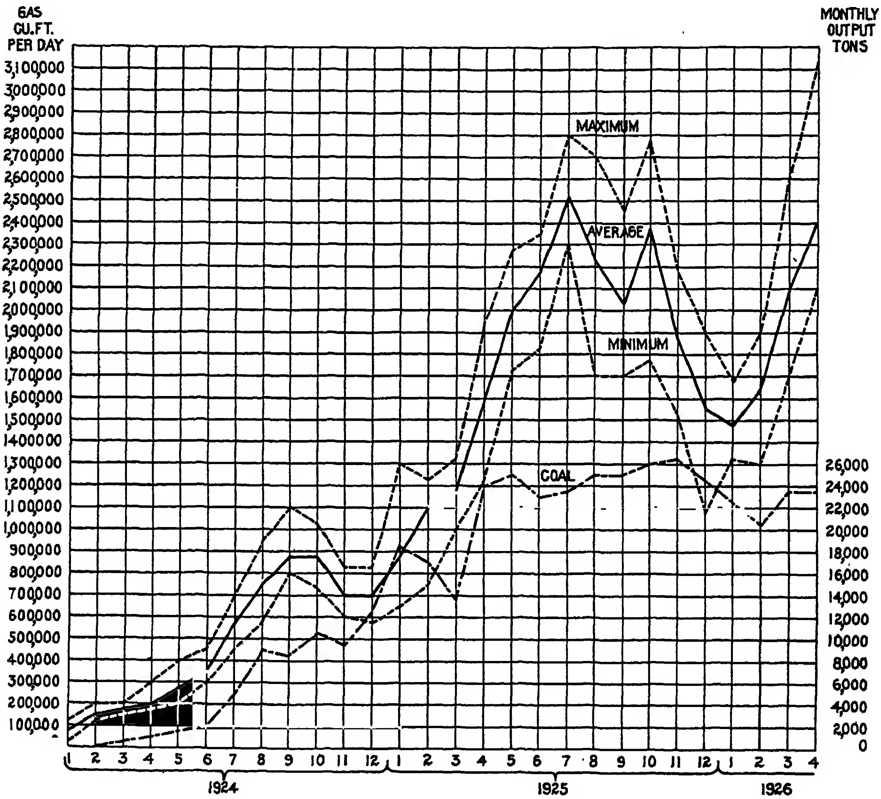

FIG. 4.—RELATION OF COAL MINED AND VOLUME OF GAS IN A MINE IN POCAHONTAS NO. 4 SEAM, POCAHONTAS REGION, WEST VIRGINIA.

In Fig. 2 are shown similar results from an adjoining mine in the same seam. The two mines are about identical as regards geological conditions and depth of coal and were opened up at the same time and maintained practically the same coal output for several years. Physical conditions are identical as nearly as can be described in these two mines, yet the charts are entirely different. In Fig. 2, the gas flow evidently has not

reached its maximum. It illustrates again that the flow does not cease with the discontinuance of coal mining.

In Fig. 3 is illustrated the case of a mine in the Pittsburg seam of the Fairmont region that is nearing exhaustion. This mine was originally opened about the year 1854, by a drift in a tract of coal of 1800 acres. In 1888, a shaft was sunk to a depth of 125 ft. in a more central location. Within the last 10 years the adjoining territory on all sides has been completely developed. The diagram indicates that a reduction in the flow of gas may be expected after the territory has been fully developed.

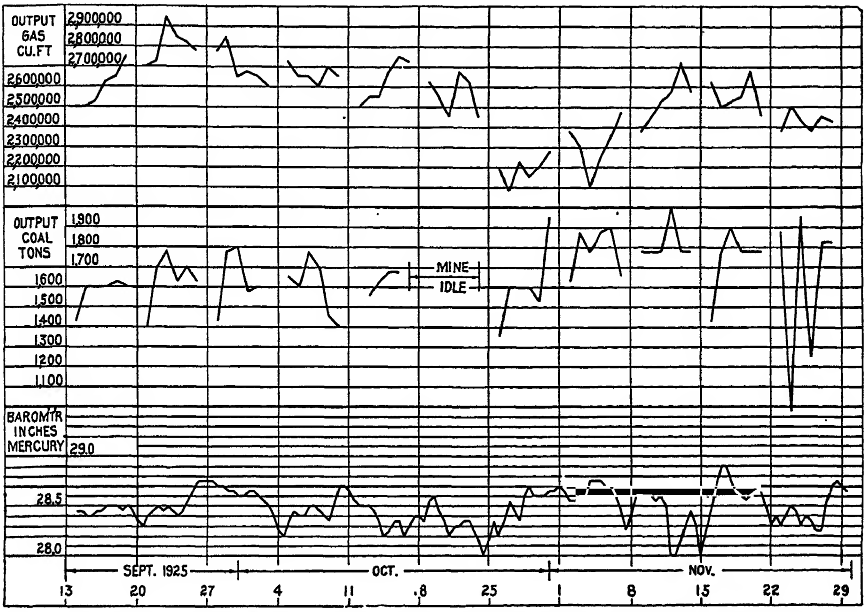

Fig. 5.—Daily fluctuation of gas with tonnage and barometer in a mine in Pocahontas No. 4 Seam, Pocahontas Region, West Virginia.

In Fig. 4 is illustrated the case of a mine in the No. 4 Pocahontas seam of coal in McDowell Co., W. Va. This is a shaft mine 565 ft. deep in virgin territory. The maximum and minimum lines indicate the highest and the lowest daily result during the month. The average indicates that during the first year the rate of flow of gas follows closely the rate of development. It also shows that the flow may fall off very materially for no perceptible reason.

In Fig. 5 daily results are plotted for a period of about 3 months with the corresponding daily output of the mine and also the record of the barometer during the same period. This is a shaft mine 560 ft. deep in the No. 4 Pocahontas seam of coal and has been operating about 10 years.

Explosibility of Coal and Other Dusts in a Laboratory Steel Dust Gallery*

By V. C. Allison,† Pittsburgh, Pa.

(New York Meeting, February, 1926)

Large-scale testing of the explosibility of coal dust as conducted by the Bureau of Mines in its Experimental Mine involves a large initial investment, and a high charge for maintenance and conduct of the test; in addition the work is necessarily slow because of the time required for the preparation of the tests and the cleaning and repairing of the mine after each test. These tests have been made along practical mining lines that admit of a range of mixtures of dust varying from 5 to 10 per cent. and with gas as low as 0.5 per cent. increments. Because of the time and expense involved, efforts have been made to devise a type of gallery for laboratory use in which the results obtained in the Experimental Mine may be correlated as an aid in a study of dusts, the explosibility of which might be approximately determined quickly and at a minimum of expense.

The explosibility of coal dust has been studied by a number of European investors, using small laboratory devices; among these may be mentioned the experiments conducted by Galloway, Vital, Hall and Clark, Abel, Mallard and Chatelier, Thorpe, Bedson and Widdas, Engler, Holwartz and Meyer, and Taffanel, results of whose experiments have been reviewed by the Bureau of Mines.[1] The Bureau has conducted many tests with the Frazer-Clement[2] apparatus, which gives the pressures set up when the dust cloud within is ignited with a platinum coil or tube, heated to different temperatures, but for these tests the sample must be especially prepared as to size (200 mesh) and must be dry.

Need for a Dust-testing Apparatus

An attempt has been made to design an apparatus that may be used for checking and correlating the experimental tests. The first design was a wooden gallery, 8 by 10 in., inside cross section, and 12 ft. long, and having a number of strips along each side for creation of turbulence. The wooden gallery did not get beyond its period of calibration; it was difficult to make the gallery sufficiently tight, for it was necessary that the top be removable in order that the gallery could be cleaned. In the use of the wooden gallery, however, sufficient operating data were obtained to be a guide in the design of a steel gallery having a circular cross-section. One of the features was a flap valve, at the closed end of the gallery, that would open inwardly thus preventing a reduced pressure within, caused by the cooling of the gases of combustion, and the explosions propagated much better than when the valve was not used; another feature was the determining of the best method for getting a cloud of dust simultaneously in all parts of the gallery, which was accomplished by the use of the concussion wave of a small quantity of powder or by the use of compressed air.

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† Assistant fuel chemist, Bureau of Mines Experiment Station.

[1] J. C. W. Frazer, E. J. Hoffman, and L. A. Scholl, Jr.: A Laboratory Study of the Inflammability of Coal Dust. Bur. of Mines, *Bull.* 50 (1913), 115.

[2] J. K. Clement and L. A. Scholl, Jr.: The Inflammability of Illinois Coal Dusts. Bur. of Mines, *Bull.* 102 (1916), 74.

The laboratory steel dust gallery is 8 in. in diameter and 17 ft. long and has automatic intake valves at the closed end and a manifold fitted with nipples through which the dust is ejected by compressed air.

The conclusions on the tests conducted in this gallery are based on an empiric formula designed from a study of the data obtained in the experimental tests when correlated with the proximate analyses of the samples of coal dust used. The results, however, are not so conclusive as to justify the substitution of the laboratory dust gallery for testing out the conditions that will promote or prevent propagation of dust explosions in operating mines. The paper is presented as a study of a laboratory device, which it is thought is not only applicable to the study of the explosibility of coal dust but also to other combustible dusts. It is hoped that its review by others may be helpful in improving its design or in formulating a less complicated formula for the interpretation of test results and also to solicit criticism of the formula presented.

J. W. Paul.
Chief of Coal Mining Investigations
Bureau of Mines.

The laboratory steel dust gallery was developed in an attempt to devise a laboratory method of testing the explosibility of coal dust that would yield results applicable to actual mining conditions, something hitherto unattained.

Many dust clouds will not explode when subjected to a moderate source of ignition,[3] because, among other things, of a high heat capacity and high ignition temperature of the dust. Such a dust cloud may, however, transmit an explosion started by a more explosive material; in coal mines, this may be a "booster charge" of methane or an overcharged or improperly placed shot. The ability to explode when exposed to a moderate source of heat energy is called "ignition;" the ability to transmit an explosion already started may be called "propagation."

A cubic inch of material in the massive form offers a surface of 6 sq. in. to the oxygen of the air at the start of combustion. The same cubic inch of material, if so crushed that it all passes through a 200-mesh sieve, offers a surface of not less than 8 sq. ft. to the oxygen of the air at the start of combustion, a ratio of 190 to 1. The dust then can enter into combustion, if it has the ability, about 190 times as fast as the same material in massive form; this produces the explosion.

Description of Laboratory Steel Dust Gallery

The laboratory steel dust gallery consists of a steel tube of 8 in. inside diameter and 17 ft. long with a manifold pipe, connected to the tube by eleven riser pipes, extending beneath the tube for the first 12 ft. The tube is equipped with a removable breech end to assist in cleaning; the manifold and riser pipes can be raised or lowered from the tube for the same object. The removable breech end contains an ignition apparatus

[3] The effect of an ignition source in causing an explosion is due to three properties of the igniting flame—temperature, duration, and volume; a source of ignition deficient in any one or two of these properties may be termed a "moderate source of ignition."

and three valves, opening inwards under slight tension, to admit air behind the flame and so prevent any drag on the flame by the cooling and contraction of combustion gases. The charge, 9.3 gm. of black rifle powder is placed in the ignition apparatus and is ignited by means of a 22-caliber black-powder blank cartridge (from which the wadding has been removed to avoid the propulsive effect) fired from a rifle mechanism. A steel disk, 7⁄8 in. in diameter and 1⁄8 in. thick, rests upon a shoulder 2 in. below the top of each riser; the dust is placed upon the disks. The rear end of the manifold is connected, through a trigger valve, to a steel bottle of 0.07 cu. ft. capacity supplied with air at a pressure of 100 lb. per sq. in. by a small compressor. The sudden release of this air causes

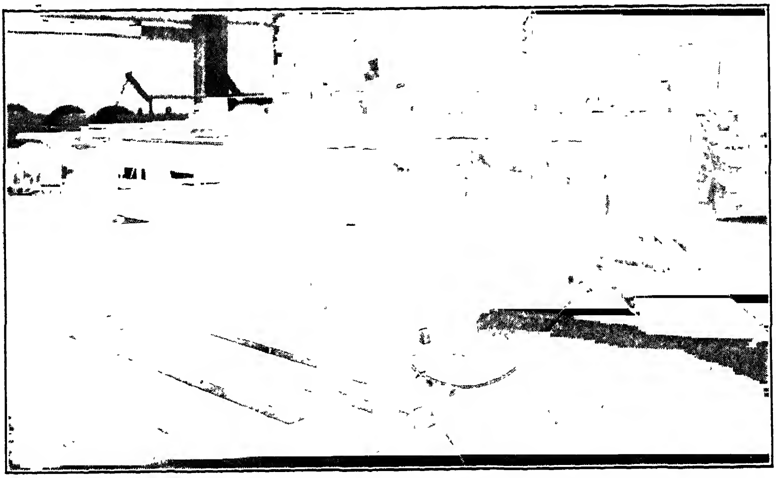


FIG. 1.—LABORATORY STEEL DUST GALLERY.

the disks and dust to rise to the tube where the disks stop, but the dust passes into the tube establishing a dust cloud in the first 12 ft. of the tube, fed with the dust in the center of each linear foot. The powder flame is fired into the dust cloud within 1 sec. after the cloud is raised and any flame length over 11.0 ft., the average flame length with no dust present, shows that the dust cloud has transmitted the flame. The flame length is measured by the burning of small pieces of guncotton, spaced every 6 in. upon a removable frame so placed that the guncotton is in the center of the tube.

The gallery is equipped with the following devices: a gas-tight parchment paper seal at the muzzle end, a gas meter for measuring the gas introduced, a circulating pump for mixing the gas and air, and a sampling device for obtaining a sample of the gallery atmosphere just previous to

firing. Fig. 1 shows the gallery loaded and ready to fire; Fig. 2 shows the rear end of the gallery with the breech open to facilitate cleaning and

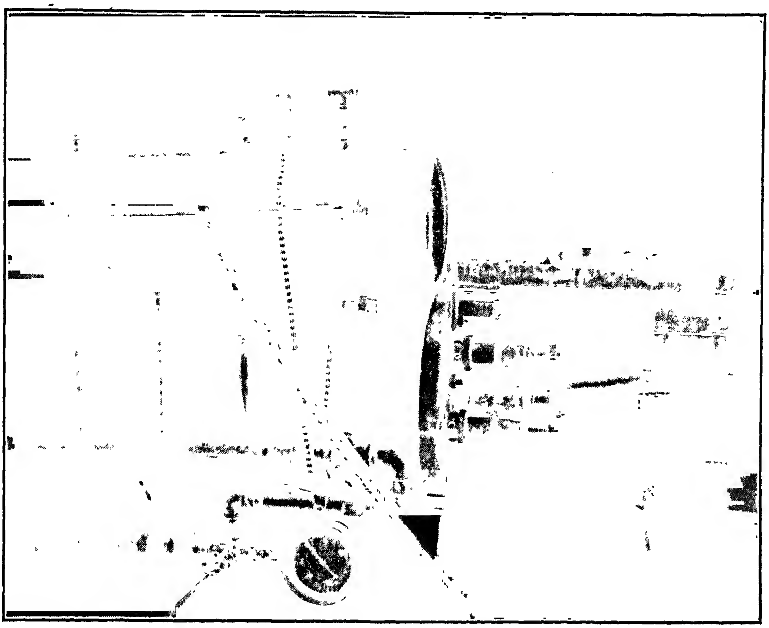


FIG. 2.—BREECH END OF LABORATORY STEEL DUST GALLERY.

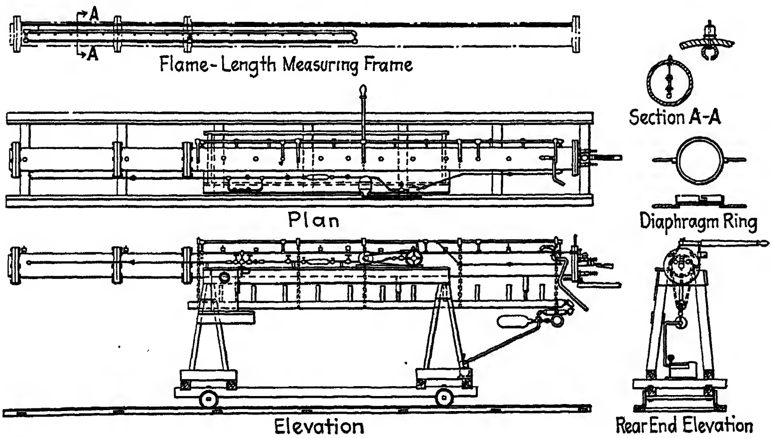


FIG. 3.—ASSEMBLY AND DETAILS OF THE LABORATORY STEEL DUST GALLERY.

the manifold lowered for reloading. The assembly of the gallery is shown in Fig. 3.

Correlation of Results of Tests

Standard Pittsburgh coal dust, from the Experimental Mine and available in unlimited quantities, was used in the first series of tests, in which 299 shots were fired, over a considerable period of time. The following purely empirical formula was devised to correlate the mass of data obtained.

$$F = \frac{3(S + 3000)(X^{1/6} - 0.00046X)}{B + 5\,\text{ash} + 12H_2O + 15CO_2 - 20CH_4}$$

in which F = flame length, in feet;
S = surface, in square centimeters, of 1 c. c. of dust;
X = concentration of dust cloud, in ounces of dust per 1000 cu. ft. of air; or in grams of dust per cubic meter of air;
Ash and H_2O = ash and moisture contents of the dust, in percentage;
CO_2 and CH_4 = contents of carbon dioxide and methane in gallery atmosphere, in percentage;
B = constant, specific to each dust.

An average shape was assumed for the dust particle in obtaining the value of S. In the case of coal dusts, if the percentage of the all-through-20-mesh dust that passes through a 200-mesh sieve is called M, $S = 7M + 230$.

Temperature, humidity, and barometric pressure had no appreciable effect upon the explosibility of coal dust.

If the ash, moisture, and size values for standard Pittsburgh coal dust are substituted in the formula and no CO_2 or CH_4 are present, the relation of flame length to concentration is,

$$F = 8.8(X^{1/6} - 0.00046X)$$

This is shown graphically in Fig. 4, which also shows the concentration effect to be practically a constant, a straight line, from 100 oz. per 1000 cu. ft. to 1000 oz. per 1000 cu. ft. The explosibility of a fine dust passes through a maximum with increasing concentration and then decreases with further increase of concentration; but this does not apply to the mixed coarse and fine dusts encountered in the mine. Such a mixed dust may be present in such large amounts that its explosibility should be reduced by this concentration effect, but the coarse dust does not readily enter into an explosion and there may be just the right amount of fine dust in the mixed dust, even though the percentage content is low, to obtain the maximum explosive concentration when raised in a dust cloud—and the fine dust is the dust first raised by an incipient explosion wave.

The flame length-concentration curves for seven type dusts, chosen by and obtained through the courtesy of the Bureau of Chemistry, are

given in Fig. 5. They all qualitatively indicate an upper and lower explosive limit similar to an explosive gas; in both cases this is due to the same effects—oxygen deficiency and the high heat capacity of the dust cloud or gas. The lower explosive limit is 34 oz. per 1000 cu. ft. of

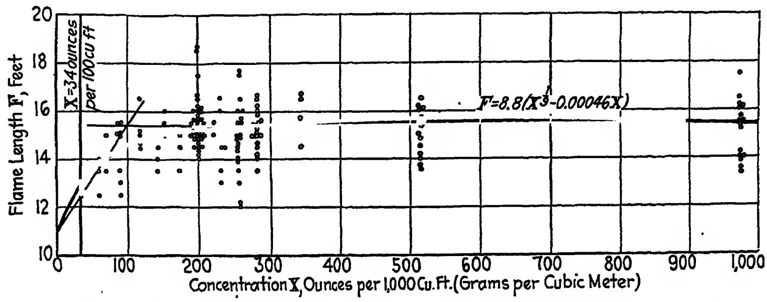


FIG. 4.—FLAME LENGTHS OF STANDARD PITTSBURGH COAL DUST WITH VARYING CONCENTRATION OF DUST CLOUDS.

air for all the dusts, but this is because the ignition course used is so intense that even shale dust, in concentrations up to 34 oz. per 1000 cu. ft. of air, will carry enough of the heat energy of the powder flame to

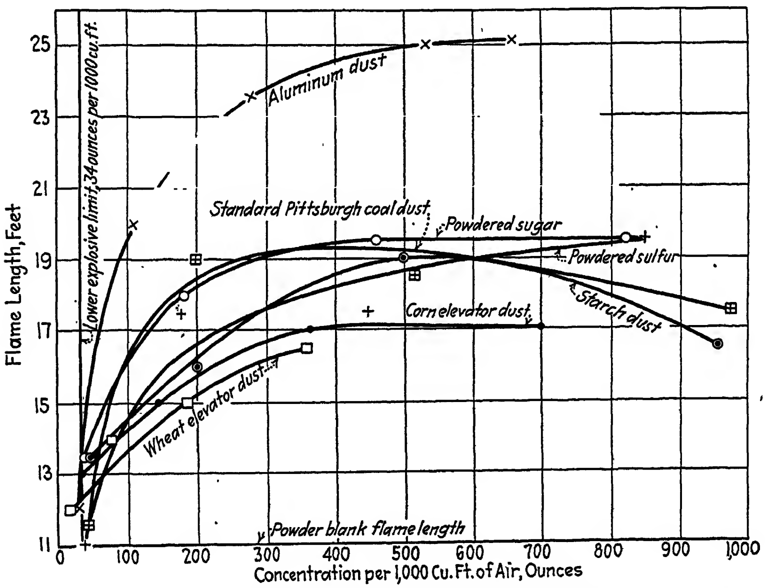


FIG. 5.—EXPLOSIBILITY-CONCENTRATION CURVES OF SEVEN TYPE DUSTS.

lengthen the flame appreciably. Flame lengths obtained at concencentrations below 34 oz. per 1000 cu. ft. in the laboratory steel dust gallery, therefore, possess no significance. It will be noticed that the

flame length is different for each dust at 34 oz. per 1000 cu. ft.; these flame lengths indicate the relative lower explosive limits.

The flame length concentration curves are not the same for different types of dust. It is possible to have two types of dust, *A* and *B*, one of which, *A*, is much more explosive than the other, *B*, at low concentrations; but the explosibility of *B* may increase with increasing concentration so much more rapidly than that of *A* that a concentration may be attained where *B* is more explosive than *A*; a third type of dust, *C*, may have a much higher explosibility at low concentrations than either *A* or *B* but its explosibility may increase so slowly with increasing concentration that its explosibility at its maximum explosive concentration will be less than that of either *A* or *B*.

There are then three characteristics of the explosibility-concentration curves that are of special interest; the lower explosive limit or initial explosibility, the rate of increase of explosibility with increasing concentration over the sharply rising part of the curve, and the maximum explosibility or explosibility at the maximum explosive concentration. Table 1, in which the different type dusts are ranked according to their explosibility, presents this discussion in a brief form:

TABLE 1

Dust	Initial Explosibility	Rate of Increase of Explosibility with Concentration	Maximum Explosibility	Maximum Explosive Concentration, Oz. per 1000 Cu. Ft.
Starch	1	6	4	200 to 600
Sugar	2	4	3	200 to 1000
Corn elevator	3	5	6	200 to 600
Aluminum	4	1	1	500 up
Wheat elevator	4	4	7	200 to 600
Coal	5	3	5	200 to 600
Sulfur	6	2	2	1000 up

As a direct application of this information to a specific dust, note the sulfur dust. It is the least explosive of the seven type dusts at low concentrations; as the concentration increases its explosibility increases at a rate only exceeded by that of aluminum dust; its explosibility at its maximum explosive concentration is second only to that of aluminum dust. An interpretation of this behavior is that sulfur dust, although somewhat more difficult to ignite at low concentrations than the other dusts, becomes very dangerous at higher concentrations.

Relative Explosibility

Substituting $X = 285$ oz. per 1000 cu. ft. of air (a representative concentration lying within the maximum explosive range) in the flame formula and stating in terms of B,

$$B = \frac{5.22(S + 3000)}{F} - (5\text{ ash} + 12H_2O + 15CO_2 - 20CH_4)$$

The probable error $\pm \frac{2}{3}\sqrt{\frac{\Sigma e^2}{n - c}}$ of the individual B value for 299 shots is ± 4.6 per cent. and the probable error of the average B value, $\pm \frac{2}{3}\sqrt{\frac{\Sigma e^2}{n(n - c)}}$, is ± 0.3 per cent. e = individual error, n = number of tests, C = constants determined.

B is a constant, specific to each dust, and the ratio of B for standard Pittsburgh coal dust to B of any other dust gives the explosibility of the dust, relative to the explosibility of standard Pittsburgh coal dust. Ninety-two samples of miscellaneous dusts were tested in the laboratory steel dust gallery (from three to seven shots each) and their B value thus determined. For standard Pittsburgh coal dust, $B = 1117$; this value divided by the B of the other dust gives the relative explosibility of the other dust. These relative explosibilities are given in Fig. 6, where standard Pittsburgh coal dust is 100 and anything over 100 is more explosive than standard Pittsburgh coal dust; a reading of 60 on this scale represents zero explosibility. The relative explosibility refers to that of the moisture and ash-free dusts, the moisture and ash effect having already been accounted for in the flame formula. It is obviously impossible to construct a relative explosibility table or chart for all the possible moisture and ash contents of the various dusts.

There are several methods for reducing the explosibility of a dust, all of which depend primarily on dilution. The explosibility of an explosive gas can be overcome by diluting with air until the gas concentration is below its lower explosive limit; the removal of methane by ventilation is an example. Or the gas concentration may be reduced, at the same time that the oxygen content of the air is lowered, by the introduction of an inert gas. The second of these methods, if possible of application, would be unsuitable in places where men must work.

The explosibility of a dust can be overcome by diluting with water or an inert dust. Coal dust is hard to wet unless it contains at least 5 per cent. moisture; however, if 20 per cent. of inert dust is previously mixed with the coal dust, the mixture wets easily. There is a danger, however, that the mixture will dry out and cake, through neglect in watering; then dry explosive dust will accumulate on top of the caked mixture—this is especially true of winter conditions when the air is dry. If rightly done, watering is probably unexcelled as a preventative of dust

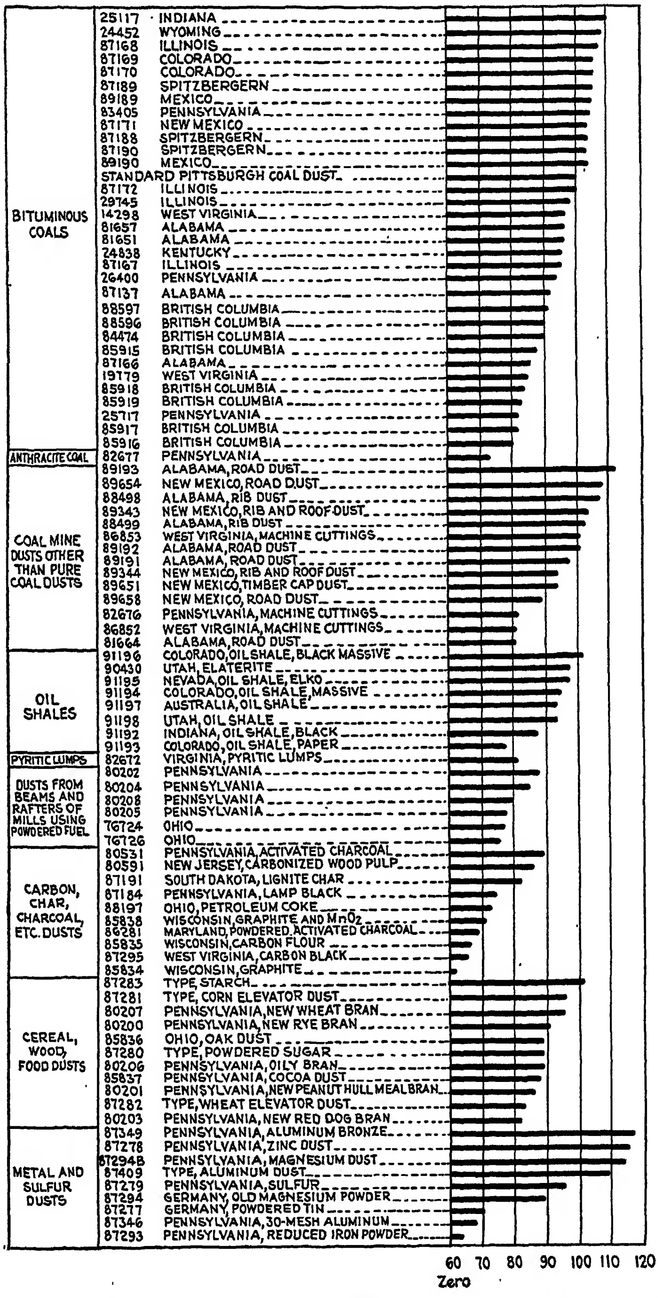


FIG. 6.—RELATIVE EXPLOSIBILITY OF COMBUSTIBLE PORTION OF 92 SAMPLES OF MISCELLANEOUS DUSTS BASED ON THE EXPLOSIBILITY OF STANDARD PITTSBURGH COAL DUST AS 100; MOISTURE AND ASH WILL DECREASE THESE VALUES; 60 ON THIS SCALE IS ZERO EXPLOSIBILITY.

explosions. The other method, not so easily affected by neglect, is to dilute the coal dust by mixing it with rock dust. The major effect of either rock dust or water in reducing the explosibility of an explosive dust lies in the heat capacity of the water or rock dust. This is shown by the fact that water is about 2.4 times as effective in reducing the explosibility of coal dust as rock dust; this is about the ratio of their heat capacities.

The gauze in a safety lamp cools the flame below the ignition temperature of methane by absorbing and conducting the heat away. The rock dust in a dust cloud composed of a mixture of rock dust and coal dust cools the flame below the ignition temperature of the coal dust by absorbing the heat.

If the value of $X = 285$ ounces per 1000 cu. ft. of air is substituted in the flame formula and this formula then solved for ash, the following arrangement is obtained:

$$\text{Ash} = \frac{1.04(S + 3000)}{F} - \frac{B}{5} - 2.4H_2O - 3CO_2 + 4CH_4$$

If sufficient inert material is present in the mixture of inert dust and explosive dust to just neutralize the explosibility of the explosive dust, the flame length of the mixture will be the same as the powder-blank flame, 11.0 ft. Substituting $F = 11.0$, the ash formula takes the form,

$$\text{Inert} = 0.095S + 283 - \frac{B}{5} - 2.4H_2O - 3CO_2 + 4CH_4$$

What is wanted is the amount of inert dust, I, that must be added to the explosive dust to make 100 parts of a non-explosive mixture of the inert dust and explosive dust. The ash already present in the explosive dust must be given credit for the part it plays in reducing the explosibility of the mixture.

The general form of the rock-dust formula, after giving credit to the moisture and ash of the explosive dust, becomes,

$$I = 0.095S + 283 - \frac{B}{5} - 3.4H_2O - \text{ash} - 3CO_2 + 4CH_4$$

As a matter of fact, the rock-dust formula that fitted the experimental data of the laboratory steel dust gallery was,

$$I = 0.06S + 170 - 0.12B - 2H_2O - 0.6\ \text{ash} - 1.8CO_2 + 2.4CH_4$$

Attention is directed to the similarity between the rock-dust formula and,

$$I = 0.6(0.095S + 283 - \frac{B}{5} - 2.4H_2O - \text{ash} - 3CO_2 + 4CH_4) \quad \text{or}$$

$$I = 0.06S + 170 - 0.12B - 1.4H_2O - 0.6\ \text{ash} - 1.8CO_2 + 2.4CH_4.$$

There is evidently a direct relation between the two and the only discordant quantity is the moisture coefficient. This may be due, in

part, to the fact that the moisture and ash in the flame formula represented the total moisture and ash of the mixture, whereas in the rock-dust formula it represents only the moisture and ash of the explosive dust which is only a part of the mixture. The factor 0.6 probably involves a number of factors not evaluated, heat effects preponderating.

The rock-dust formula applying to laboratory steel dust gallery conditions,

$$I = 0.06S \div 170 - 0.12B - 2H_2O - 0.6 \text{ ash} - 1.8CO_2 + 2.4CH_4$$

was tried out on 77 of the 92 miscellaneous dust samples by actually determining the I values required and comparing these with the calculated I values. Of the 77 samples of dust, 68 were mixed with the calculated I quantity of rock dust and only 6 of these mixtures gave flame lengths greater than the powder-blank flame range. The remaining 9 samples were mixed with 8 parts less of rock dust per 100 parts of the mixture than the calculated I quantity and all of these mixtures gave flame lengths greater than the powder-blank flame range. Eight samples of coal dust were chosen and from four to six mixtures with rock dust, on either side of I, were tested out in the laboratory steel dust gallery. The flame length of the mixture shortened and approached the powder-blank flame range as I was approached and the flame length of the mixture became less than the powder-blank flame range when the I quantity of rock dust in the mixture was exceeded. This happened with each of the eight samples. Ten samples of coal that had previously been thoroughly tested at the experimental mine were obtainable. The rock-dust formula for Experimental Mine conditions is,

$$I = 0.05S + 142 - 0.1B - 1.7H_2O - 0.5 \text{ ash} - 1.5CO_2 + 2CH_4$$

Attention is called to the similarity of this rock dust formula to,

$$I = 0.5\left(0.095S + 283 - \frac{B}{5} - 2.4H_2O - \text{ash} - 3CO_2 + 4CH_4\right)$$

or

$$I = 0.05S + 142 - 0.1B - 1.2H_2O - 0.5 \text{ ash} - 1.5CO_2 + 2CH_4$$

The comments as to the relation between this rock-dust formula and the flame formula are the same as in the case of the rock-dust formula for the laboratory steel dust gallery conditions. The difference between the factors 0.6 and 0.5 probably represents a number of things, such as size, shape, and length of gallery, and gallery wall material; the size factor probably predominates.

The rock-dust formula for Experimental Mine conditions is then,

$$I = 0.5S + 142 - 0.1B - 1.7H_2O - 0.5 \text{ ash} - 1.5CO_2 + 2CH_4$$

Sixteen tests were run upon 10 samples of coal, 6 with CH_4 from 0.78 per cent. to 2.0 per cent., and the average discrepancy between the

actual and the calculated I values was ± 4 parts of inert dust in 100 parts of the mixture of explosive dust and inert dust.

Volatile matter is not a necessary constituent of an explosive dust, as shown by the explosibility of tin, aluminum, magnesium, and zinc dusts and by finely divided reduced iron; their relative explosibilities are given in Fig. 5. The presence of volatile combustible matter in a dust is, however, known to increase the explosibility of that dust.

There are two explosive factors in a dust: the first is the explosibility of the dust as a dust; and the second is due to explosibility of that part of the volatile matter made available by the effects of the explosion flame. Apparently the integral effect of the 1 gm. of 60-mesh coal in a more or less compact mass, the 950° C. temperature, and the 5-min. interval of the official method of determining the volatile matter, is proportional to the integral effect of the suspended fine dust, the temperature of the explosion flame, and the brief duration of the explosion flame in the laboratory steel dust gallery. The ratio of the volatile matter to the total combustible of a dust is an indication of the explosibility of that dust. The quantity B in the flame and rock-dust formulas is an inverse quantity; the larger B, the smaller is the explosibility. The $\frac{V}{V + FC}$ increases the explosibility of a dust so it must be subtracted from that part of B which expresses the explosibility of the dust as a dust. B can thus be replaced in the rock-dust formula applying to laboratory steel dust gallery conditions by,

$$1460 - 868\left(\frac{V}{V + FC}\right) \text{ which gives,}$$

$$I = 0.06S + 104\left(\frac{V}{V + FC}\right) - 2H_2O - 0.6\,\text{ash} - 1.8CO_2 + 2.4CH_4 - 5$$

This formula enables the calculation of I, under laboratory steel dust gallery conditions, from the proximate and size analyses alone.

The explosibility of a fine coal dust is favored by its volatile ratio and reduced by its moisture and ash content. The I value for a fine coal dust can thus be predicted from its volatile ratio and its moisture and ash content. The miscellaneous cereal and wood dusts have a volatile ratio of approximately 0.8; the I value for these dusts can therefore be predicted from their moisture and ash content alone. The oil shales also have a very high volatile ratio and, in addition, contain so little moisture that their I value can be calculated from their ash content alone. The formulas for calculating the I values for these dusts, under conditions similar to those of the laboratory steel dust gallery, are as follows:

Powdered starch Corn-elevator dust	$I = 97 - 2(H_2O + \text{ash})$
Powdered sugar Wheat-elevator dust Cocoa dust	$I = 80 - 2(H_2O + \text{ash})$

Wood dust (oak)........................	$I = 45 - 2(H_2O + \text{ash})$
Brans..................................	$I = 30 - 2(H_2O + \text{ash})$
Oil-shale dusts........................	$I = 110 - \text{ash}.$

When using an inert gas to reduce the explosibility of a dusty atmosphere (not applicable to processes where men work in the dust), 7.5 parts of rock dust is subtracted from I for each per cent. of oxygen deficiency (each per cent. below 21 per cent.) in the atmosphere. For example, starch (containing 11.2 per cent. H_2O and no ash) would require an I of 97–2 (11.2 + 0) or 75 parts of rock dust, 75/7.5 = 10; so 10 per cent. oxygen deficiency in the dust atmosphere would prevent starch from exploding. Starch would be non-explosive in an atmosphere containing 10/21, or 48 per cent. inert gas and 52 per cent. normal air.

The B value in the rock-dust formula relating to Experimental Mine conditions can be replaced by $1420 - 790\left(\frac{V}{V + FC}\right)$ and this gives,

$$I = 0.05S + 79\left(\frac{V}{V + FC}\right) - 1.7H_2O - 0.5\ \text{ash} - 1.5CO_2 + 2CH_4$$

The average discrepancy between the I values actually required at the Experimental Mine, on the 16 tests on the 10 samples of coal, and the I values calculated from the above formula is ± 4.2 parts of inert dust.

The average S value for 32 samples of bituminous-coal dusts and 1 sample of anthracite dust, all ground through 200 mesh, was 920; $0.05S$ is then $0.05 \times 920 = 46$. Substituting 46 for $0.05S$ in the rock-dust formula relating to Experimental Mine conditions,

$$I = 46 + 79\left(\frac{V}{V + FC}\right) - 1.7H_2O - 0.5\ \text{ash} - 1.5CO_2 + 2CH_4$$

If I so calculated is less than 60, subtract 5; if I so calculated is greater than 60, add 5.

The I values calculated by means of this formula, on the 16 tests on the 10 samples of coal, show an average discrepancy over the actual I values of ± 6.2 per cent. or a probable error, $\pm \frac{2}{3}\sqrt{\frac{\Sigma e^2}{n - c}}$, of the same amount; and ± 6.2 per cent. of a mixture containing $I = 67$ is ± 4.1 parts of inert dust.

The formula,

$$I = 46 + 79\left(\frac{V}{V + FC}\right) - 1.7H_2O - 0.5\ \text{ash} - 1.5CO_2 + 2CH_4,$$

affords the means of calculating the amount of rock dust necessary to mix with the fine dust from a coal to render it non-explosive under experimental-mine conditions, from the proximate analysis alone, with an error not greater than ± 4 parts of the rock dust. The results upon the 16 tests on the 10 samples of coal dust are shown in Fig. 7. These results

are for fine dusts. The ordinary mine dusts are not quite so fine and, therefore, if the amount of mine dust present is not large, 17 parts of the rock dust may be subtracted from I. When the amount of dust present is large, the actual amount of fines present may be great enough to set up the maximum explosive concentration even though the percentage of the fines present in the dust is small. The explosion of a small pocket of methane may start a feeble explosion; such an explosion wave will stir up the fines before the coarse particles.

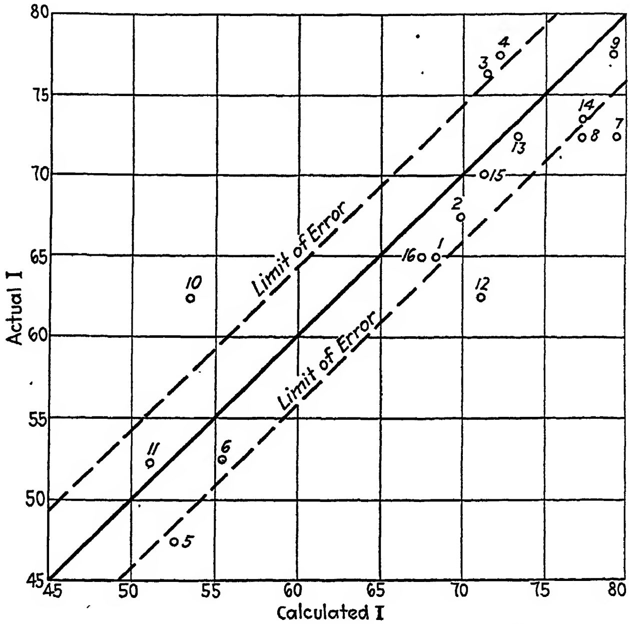


FIG. 7.—CALCULATED AND ACTUAL I VALUES FROM 16 TESTS WITH 10 SAMPLES OF COAL DUST.

As an example of the application of the rock-dust formula, suppose that the I value for a sample of road dust from a certain mine is desired. The composition of the face coal from this mine is,

CONSTITUENT	PER CENT.
H_2O	3.0
Volatile	38.0
Fixed carbon	54.0
Ash	5.0
Total	100.0
$V/(V + FC)$	0.350

I for the face-coal dust is therefore obtained as follows:

$$I = 46 + 79 \times 0.35 - 1.7 \times 3.0 - 0.5 \times 5.0 - 0 + 0 = 66$$

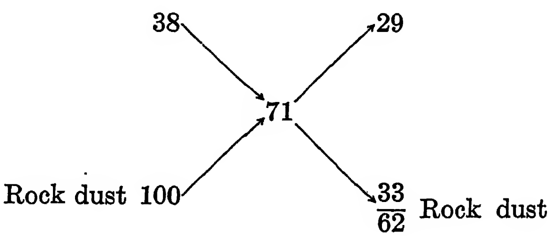
As 66 is greater than 60, 5 is added so that $I = 71$.

The composition of the road dust is.

Constituent	Per Cent.
H_2O	5.0
Volatile	23.0
Fixed carbon	32.0
Ash	40.0
Total	100.0

In the I formula, the moisture and ash in a coal dust are given credit for the part they play in reducing the explosibility of the coal dust. The moisture and ash in the face-coal dust is "native;" the moisture and ash in a mine dust sample from the same mine, which is in excess of the native moisture and ash of the face coal, is "added" ash and moisture—generally from the roof and bottom. The moisture, whether native or added, has the same effect in stopping an explosion. The moisture credit due a mine dust is, from the I formula, 1.7 times the difference between the moisture in the face-coal dust and the moisture in the mine dust; in this case it is 1.7 times $5 - 3$, or 3.4. The added inert material (customarily determined as ash) is more effective in stopping an explosion than the native ash of the face-coal sample, which native ash effect is already accounted for in the I formula. The ash credit of a mine dust over the face-coal dust is then the difference between their two ash contents; in this case $40 - 5 = 35$. The total credit (in stopping an explosion) of this road dust is then $35 + 3.4 = 38.4$ or 38.

The following inverse proportion device is now used to find how much rock dust must be added to the road dust to make it non-explosive. The road dust, because of its large moisture and ash content, has an explosion-stopping credit of 38 and rock dust has an explosion-stopping power of 100; $I = 71$ for the face-coal dust.



The road dust then requires an I value of ${}^{33}\!/_{62}$, or 53.

The total inert content of the non-explosive rock-dust, face-coal dust mixture is $71 + 0.29$ times the sum of the moisture and ash content of the face coal, or 73 per cent. The total inert content of the non-explosive rock-dust, road-dust mixture is $53 + 0.47$ times the moisture and ash content of the road dust, or 74 per cent. If the road dust is not present

in too large quantities (if the mine is kept well cleaned), 17 may be taken away from the I value; $I = 53 - 17 = 36$.

Coal from the Experimental Mine that had been ground in a ball-mill was considerably more explosive than coal from the same mine that had been ground in a Raymond impact mill using a jet of air to separate the fines from the coarse particles. The ultimate and proximate analyses and the B.t.u. showed no difference to account for this effect; so 50 gm. of each dust was connected to a vacuum pump and all the gas that would come off at room temperature was removed. Each dust yielded 1.176 c. c. of gas per gram of dust. A noticeable difference was apparent in the composition of the two samples of gas.

Standard Pittsburgh Coal Dust Ground in a Raymond Impact Mill, Per Cent.	Constituent	Same Coal Ground in in a Ball-mill, Per Cent.
2.04	Carbon dioxide	2.78
3.16	Oxygen	2.80
trace	Carbon monoxide	0.10
0.19	Total hydrocarbons	0.34
0.00	Hydrogen	0.00
94.61	Nitrogen	93.98

The coal ground in the ball-mill was acted on only by the oxygen in the charge of air present in the mill when the jar was sealed. Everything in the gas analyses points to an active oxidation proceeding in the ball-mill dust; this oxidation had already taken place in the Raymond mill dust, presumably by the air used in separating the fines from the coarse particles. Some easily oxidized constituent was affected, which greatly lowered the explosibility of the dust but did not affect the heat units noticeably; this ingredient is present only in small quantities. This may be of considerable importance in the pulverized-fuel industry and suggests the part played in the explosion of a dust by those constituents that are appreciably volatile at ordinary temperatures; this applies especially to freshly made dusts.

Conclusions

1. The I value of a fine coal dust for experimental mine conditions is given by,

$$I = 46 + 79\left(\frac{V}{V + FC}\right) - 1.7H_2O - 0.5\,\text{ash} - 1.5CO_2 + 2CH_4,$$

with an average error of ± 4 parts of rock dust. For a coarse, mixed, dust, not present in too great a quantity, subtract 17 parts of rock dust from I.

2. The term CO_2 in the rock-dust formulas can be replaced with $7.5 \times$ oxygen deficiency, when dealing with an inert gas as the explosion preventative.

3. Any dust not already in its most highly oxidized condition may explode if divided finely enough.

4. The explosibility of a dust can be divided into three phases: the intitial explosibility, the rate of increase of explosibility with increase of concentration, and the maximum explosibility.

5. The explosibility of a dust can further be divided into two parts: (1) the explosibility of the dust as a dust, and (2) the explosibility of that part of the dust made more available by the explosion flame *i. e.* the volatile matter.

6. The standard official method of determining the volatile matter of a coal dust gives an indication of the explosibility of the dust.

7. Combustible volatile matter is not necessary in an explosive dust; when present, however, it increases the explosibility of the dust.

8. Moisture is about 2.4 times as affective as the ash content in reducing the explosibility of a dust; this is about the ratio of their heat capacities.

9. The explosibility of a dust has no necessary connection with the rate of combustion of the dust; the time element in an explosion is not great.

Acknowledgments

This work was done under the general direction of Mr. J. W. Paul and Mr. G. S. Rice of the U. S. Bureau of Mines. The author wishes to especially express his gratitude to Mr. A. C. Fieldner and Mr. C. M. Bouton, of the Pittsburgh Station of the Bureau of Mines, for their constructive assistance in arranging the method of presenting the work.

State Coal Mining Laws Concerning Ventilation

By John A. Garcia,* Chicago, Ill.

(Pittsburgh Meeting, October, 1926)

A standard set of coal mining laws for the entire United States is hardly practicable, yet the numerous variations in the state laws for almost every item seems entirely unnecessary. The same useless variations appear in almost every comparable section with no apparent reason except that no attempt has ever been made to codify them, and as they were developed over a long period of time by innumerable individuals and legislatures, the present situation is the logical outcome of lack of cooperation between states.

The ventilation of a mine is affected indirectly by many provisions not listed specifically under this heading, so that it would be necessary to tabulate the major portion of all the laws to secure a complete comparable digest; therefore only a few of the more important have been set down here. This point is well illustrated in the section providing for clearance between car and rib, which is intended as a safety measure against personal injury, but is very important from the standpoint of ventilation because of choked airways in mines on two-entry system.

Confusing Phraseology

Misinterpretation and confusion may be expected from the phraseology alone; one state provides that the scale of the mine map shall be "not more than 100 ft. to 1 in.;" another state demands "not less than 100 ft. to 1 in." Both probably mean the same thing and seek to prevent the operator from furnishing a map on which the workings will appear so small that the detail is illegible. Legislation is certainly indefinite that specifies structures must be fireproof when "near" a mine opening. and just what is a "dusty" or "gaseous" mine is open to argument in most of the states.

The variation in spacing crosscuts and break-throughs and the limited number of men on a split seem entirely unnecessary, as the distance is not reduced in gaseous fields, excepting in Arkansas and Tennessee, which states produce a comparatively small amount of coal, and the number of men differs 100 per cent. and more for no apparent reason.

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* Consulting mining engineer.

DIGEST OF LAWS CONCERNING COAL-

	Minimum Amount of Air Required Per Man, Per Min., Cu. Ft.	Type of Fan and Use of Closed Lights	
		Must Fans Be Reversible?	Must Safety Lamps Be Used Exclusively?
Bureau of Mines Operating Regulations.	100.	Yes	Yes, in gaseous mines or gaseous sections of a mine.
Alabama	100.	No	Not mandatory.
Arkansas	200.	Yes	Not mandatory.
Colorado	100.	Yes	At judgment of state inspector.
Illinois	150 in gaseous mines, 100 in others.	No	When necessary in judgment of mine manager or state inspector.
Iowa	100.	No	Not mandatory.
Indiana	100.	Yes	At discretion of state inspector.
Kentucky	100.	No	Mandatory in gaseous mines.
Kansas	100 where coal is 3 ft. or more thick, more for thinner seams.	No	Not mandatory.
Missouri	100.	No	Not mandatory.
Montana	150 in gaseous mines, 100 in others.	No	Mandatory in gaseous mines.
Michigan	200 in gaseous mines, 100 in others.	No	Mandatory in gaseous mines.
New Mexico	100.	No	Law reads "when necessary."
Oklahoma	200 in gaseous mines, 150 in others.	No	No provision.
Ohio	150 in gaseous mines, 100 in others.	No	No provision.
Pennsylvania	200 in gaseous mines, 150 in others.	No	Yes, in gaseous mines or sections where gas is found.
Texas	100.	No	At discretion of state inspector.
Tennessee	150 in gaseous, 100 in dusty and 85 in other mines.	No	In gaseous mines at discretion of state inspector.
Utah	150 in gaseous mines, 100 in others.	Yes	Mandatory, all mines.
Virginia	150 in gaseous mines, 100 in others.	No	Mandatory in gaseous mines.
West Virginia	150 in gaseous mines, 100 in others.	No	Mandatory in gaseous mines.
Washington	100.	No	Mandatory in gaseous mines.
Wyoming	150.	No	Not mandatory.

MINE VENTILATION IN UNITED STATES

Scale of Mine Maps and Data That Must Be Shown

Scale, Ft. per In.	Direction of Air-Current Must Be Shown by Arrows?	Location Required of Doors, Overcasts, Etc.?	Location Required of Oil and Gas Wells?
Not more than 200 ft. to 1 in.	Yes	Yes; also firewalls, dams, squeezes, etc.	Yes
100 ft. to 1 in. or 200 ft. to 1 in.	Yes	No	Yes
Not specified.	Yes	Yes	No
Not less than 100 ft. to 1 in. nor more than 200 ft. to 1 in.	Yes	Yes, also firewalls, dams, squeezes, etc.	No
Not less than 200 ft. to 1 in.	Yes	No	Yes
Not more than 200 ft. to 1 in.	Yes	No	Yes
Not less than 200 ft. to 1 in.	No	Yes	No
Not more than 100 ft. to 1 in.	No	No	No
Not more than 100 ft. to 1 in.	No	No	No
Not specified.	No	No	No
200 ft. to 1 in. or larger.	No	No	No
Not more than 200 ft. to 1 in.	Yes	No	No
100 ft. to 1 in.	No	No	No
No smaller than 200 ft. to 1 in.	Yes	No	No
Not less than 200 ft. to 1 in.	Yes	No	Yes
Not less than 200 ft. to 1 in.	Yes	No, but require location of each split.	Yes
100 ft. to 1 in.	No	No	No
100 ft. to 1 in. or 200 ft. to 1 in.	Yes	Yes; also all ventilating apparatus, elevations, etc.	No
100 ft. to 1 in. or 200 ft. to 1 in.	Yes	No	No
100 ft. to 1 in. or 200 ft. to 1 in.	Yes	No	No
100 ft. to 1 in. or 200 ft. to 1 in. or 300 ft. to 1 in.	Yes	No	No
100 ft. to 1 in.	No	No	No
Not exceeding 200 ft. to 1 in.	Yes	No	No

DIGEST OF LAWS CONCERNING COAL-MINE

Legislation	Fireproof Construction Required	Shot Firers Required
Bureau of Mines Operating Regulations.	At mines designed to employ more than 200 men, shafts, buildings and bottom within 200 ft. of each opening, main-line stoppings, seals, escape doors, etc., must be fireproof.	Yes, gaseous mines or at discretion of superviser; also may fire shots electrically from surface.
Alabama	Boiler and engine house, also underground stables and stoppings on main slopes.	No
Arkansas	Not required	No
Colorado	All surface buildings within 300 ft. of mine opening, also permanent stoppings, overcasts, etc.	Yes, except in longwall.
Illinois	All surface buildings within 100 ft. of mine opening, also slopes, shafts, stables and mine bottom within radius of 300 ft. of shafts.	Mandatory where more than 2 lb. powder is used or where gas is generated in dangerous quantities.
Iowa	All surface buildings between mine openings. Boiler and engine houses must not be placed within 60 ft. of openings. Underground stables.	"Shot examiners" where coal is shot from the solid.
Indiana	No inflammable buildings near shafts. No boiler house within 35 ft. of opening.	Mandatory in all mines using more than 2 lb. powder or where gas is generated in dangerous quantities.
Kentucky	Not required	In gaseous mines where ordered by state inspector.
Kansas	Bathhouse fireproof construction.	Mandatory, all mines.
Missouri	Underground stables fireproof.	Mandatory in gaseous or dusty mines or where coal is shot from the solid.
Montana	Underground stables, permanent stoppings and overcasts; also no inflamable building between shaft on surface.	No
Michigan	Surface buildings within 100 ft. of fan and also underground stables.	No
New Mexico	Fan on surface and also brattice cloth, overcasts and doors must be painted with fireproof paint or covered with metal.	Mandatory, all mines.
Oklahoma	Partition between air shaft and stairway; also main stoppings and overcasts.	Mandatory, all mines.
Ohio	Boiler house must be more than 60 ft. from mine opening. Underground stables.	No provision.
Pennsylvania	All inside buildings, such as stables, pumprooms, etc., and main stoppings in gaseous mines.	In gaseous mines or sections where gas may be detected by safety lamp.
Texas	Underground stables only.	No provision.

Ventilation in United States.—(*Continued*)

Number of Men on Split and Spacing of Crosscuts		Definition of Gaseous Mines
Men per Split	Distance between Crosscuts or Break-throughs	
75	In accord with state laws but not more than 100 ft.	An entire mine is rated as gaseous when 0.5 per cent. or more of methane is found in more than one split.
No provision	Not more than 70 ft.	When gas exists in quantities sufficient to ignite or explode . . . where gas exists in dangerous quantities.
50	30 ft. in gaseous mines and 40 ft. in others.	Not defined.
Old mines, 100 New, 65	60 ft.	No fixed percentage of gas but generally where found in dangerous quantities.
100	60 ft.	Judgment of mine manager or state mine inspector.
80; by special permission, 130.	60 and 70 ft. in entries. Break-throughs must be at least 20 sq. ft. in area and crosscuts 25 sq. ft.	Not defined. Probably in judgment of mine inspector.
75	45 ft.	When state inspector finds explosive gas liberated in dangerous quantities.
60	60 ft. but inspector may allow 90 ft. in special cases.	A mine in which the percentage of explosive gas exceeds 0.75 per cent. at return of any one split in a dusty mine and exceeds 1.75 per cent. at return of any one split non-dusty mine.
Mine must have 4 splits	40 ft. and at least 21 sq. ft. in area.	Not clearly defined; mainly judgment of state inspection.
50	50 ft.	Not defined. Probably in judgment of mine inspector.
100	Crosscuts in entries 60 to 100 ft. break-throughs in rooms may be 50 to 80 ft.	Not defined. Probably in judgment of mine inspector.
100	60 ft.	Mine that generates explosive gas of a sufficient quantity to be detected by ordinary safety lamp.
No provision	No provision.	Mine that vents gas which in combination with air will induce or maintain an explosive condition.
45	30 ft. in gaseous mines an 40 ft. in others.	Mines where fire damp is, has or shall be generated in sufficient quantities to be detected by ordinary safety lamp.
No provision	Crosscuts in entries 60 ft. and break-throughs in rooms 40 to 80 ft.	Nearest definition is "A mine generating fire damp so as to be detected by a safety lamp."
70, but 90 by special permission.	Not more than 105 nor less than 48 ft.	Where explosive gas is being generated in such quantities as can be detected by an approved safety lamp; also in pillar workings.
100	Not more than 60 nor less than 30 ft.	When the inspector shall find fire damp is being generated so as to require the use of safety lamps.

DIGEST OF LAWS CONCERNING COAL-MINE

Legislation	Fireproof Construction Required	Shot Firers Required
Tennessee	Not required	Not mandatory but state inspector and company officials formulate rules governing shooting.
Utah	No inflammable structures on surface near mine entrances. Fans, also escape doors to adjacent mines.	Mandatory, all mines.
Virginia	Not required	No
West Virginia	All surface buildings near mine openings, fan building and underground stables.	No
Washington	No inflammable building on surface within 75 ft. of mine opening except tipple. Underground stable and main stoppings.	Mandatory in gaseous mines.
Wyoming	Not required	"Shot inspectors" must be employed at discretion of state inspection department.

VENTILATION IN UNITED STATES.—(*Continued*)

Number of Men on Split and Spacing of Crosscuts		Definition of Gaseous Mines
Men per Split	Distance between Crosscuts or Break-throughs	
50	Gaseous mines: crosscuts 60 ft.; break-throughs, 75 ft. Dusty mines: crosscuts 67½ ft.; break-throughs, 82½ ft. Other mines: crosscuts 75 ft.; break-throughs, 90 ft.	Any mine liberating sufficient fire damp to be detected on the flame of a "modern lamp" may be so classed in judgment of inspector.
75	Crosscuts 75 to 200 ft., break-throughs 50 to 100 ft.	When inflammable gas can be detected by approved safety lamp in any working place over certain time period, or such place shows air to contain 1 per cent. or more, or if 0.5 per cent. or more in any return current or abandoned working not sealed. Also in places where there is liable to be inrush of gas due to falls.
60 to 80	Not to exceed 80 ft.	Practically in judgment of state mine inspector.
60 to 80	Not to exceed 80 ft.	Opinion of the Department of Mines.
70 to 90	Not more than 60 ft.	Practically in judgment of state mine inspector.
50	Crosscuts 100 ft.; break-throughs 50 ft.	Where gas is found and at discretion of mine inspector.

Confusion of Laws in Same Competitive Area

One of the main difficulties created by this condition of the mining laws is the confusion in the construction, development or operation of mines by companies having properties in several states. The construction plans for a modern mine to be built in one state cannot be used in another, although the mining conditions may be alike and often the only distinction is that the property is divided by a state line. As an example of this confusion may be cited the fact that 60-ft. crosscuts are ample in Illinois, whereas a competition mine across the line in Indiana is penalized on yardage and stopping costs because of the 45-ft. crosscut law in that state. In the same competitive area, one operator must build a fireproof plant while his neighbor, using wood, may build a duplicate mine, and ship his coal from the same seam to the same market and at the same freight rate but at a greatly reduced fixed charge for capital expenditure, especially when the laws differ as to wood-lined or fireproof shafts.

Very few of the states have any laws providing for the number of airways. An exception is Oklahoma, which has this odd law on its statute books:

> The work in all coal mines operated on room-and-pillar plan shall be prosecuted in the following manner and *none other*—to wit: Two entries must be driven parallel for ingress or egress of the air . . .

A literal interpretation of this clause would prevent one using a three or more entry system, but it is presumed the intent was to prevent single entry operation. Only in a few states do the laws provide for such vital questions as pressure or exhaust ventilation, use of booster fan, percentage of methane, sealing old works, limitation of number of doors, minimum area of airways, etc., and it would be very difficult to frame laws broad enough and yet properly specific for general application.

To secure the few data presented herein it was necessary for the writer to read, several times, most of the coal-mining laws of each state, and his general impression is that codification would not be too difficult, that it is practicable and that it is very desirable from almost any viewpoint.

DISCUSSION

E. A. Holbrook, State College, Pa.—To understand rightly the present differences in coal mining regulations in the different states, it is necessary to get a historical background and to follow the development of these operating regulations, the primary purpose of which is "the safety, health and welfare of the miner," and not the cheapest working of the mine. Especially is it necessary to understand that laws of this character are comparatively recent in origin and born mostly in the midst of industrial and legislative struggle. Therefore, it is not surprising that they lack the uniformity and definiteness of the older civil and property

codes. It is even to be questioned whether or not industrial regulations of this character should be passed upon and fixed by a state legislature as law, or whether they should take the form of orders from an intelligent state commission, and thus be amenable to change as practice and conditions change.

In England, for 20 years previous to 1850, a great struggle had been going on between the coal miners and coal-mine operators, the men demanding greater safety through better underground conditions and standards of safety maintained by government inspection and regulation. The owners contended that this interfered with their property rights and with their right and freedom to contract with their individual workmen. Finally, the men gained the right of governmental inspection, and from year to year following there were built up regulations or minimum standards of safety in ventilation and other hazards. These standards were brought together and revised into a general code in 1872. In 1911–12, another revision was made, when many provisions were added, which especially strengthened the central governmental authority.

In the United States, with the great development of anthracite mining during 1850–70, there came an influx of English, Welsh, and Scotch miners who had been through the struggles in Great Britain and who had become accustomed to working under operating laws and governmental inspection there. As a result of their efforts the Legislature of Pennsylvania, in 1869, passed a law (the first in America) providing for state inspection of the anthracite mines and a set of operating regulations which were broadened in 1870 and 1871. These laws were based on the English code then in existence.

The miners' efforts to gain state inspection spread to the bituminous field; Ohio, in 1874, provided for inspection and a code for bituminous mining. In 1877, bituminous inspection and a code of law were provided in Pennsylvania. Then state after state fell into line until every state where coal is mined in any quantity provides for some inspection and a set of safety and operating regulations. Practically all of these codes are based on and follow, in a general way, the Pennsylvania bituminous code, which was based on the anthracite and the English codes.

Mine operating regulations were not originated by trained engineers who had worked out and agreed as to what was best and safest as based on statistics and engineering data; they are mostly the proposals and demands of the miners which were modified by the demands of the operators, and then sometimes altered by the political exigencies of the legislators. Also, additions and changes have not generally come through engineering analysis, but as the result of some unusual accident or disaster. For example, the Cherry disaster in Illinois, in 1910, added severe regulations to the Illinois code, which are absent in the code of Indiana, the neighboring and competitive state. In some states, in spite

of the changes and advancements in mining practice during the last 10 years, little or no revision has been made in the mining regulations of these states. Apparently changes proposed by the miners are opposed by the operators, and vice versa.

When one inquires locally why the coal-mining regulations of that particular state are different from those of some other state, the common answer is "Oh, our mining conditions are different." As a matter of fact, coal-mining conditions basically are surprisingly alike in the various districts of this country and I see no reason, from an engineering standpoint, why a uniform coal-mining regulation, or code, would not be applicable to all states and districts. The best code I have examined is that of the U. S. Bureau of Mines called "Operating Regulations to Govern Coal Mining Methods and the Safety and Welfare of Miners on Leased Lands on the Public Domain." These government-leased mines are in several states, the seams are of different thickness, and pitch, gas and other operating hazards exist which are very different in the different districts. Yet this one code is applicable to all. The Bureau's code contains only about 19,000 words, yet it covers adequately all the provisions to be found in certain state codes that contain more than 30,000 words. In addition, the Bureau's code contains regulations not generally found in these state codes.

I do not believe that a mining-operating code should be a body of law, each provision of which is inflexible until repealed by a state legislature that presumably does not understand mining. I like the plan in operation in Utah and one or two other states. A state commission is charged with the inspection and regulation of mining. After due consultation with miners, operators, and others interested, it issues formal orders containing the requirements that mining companies and employees must meet and follow. Its powers are so broad that its orders have all the authority of the mining laws of the other states. However, as conditions change, it is comparatively simple to alter the Commission's orders. Examination shows Utah's orders are advancing steadily toward greater safety, and that they are the most logical and orderly of any of the state mining codes.

Mr. Garcia by showing in his paper the confusion in state regulations, has brought out clearly the need for a model or standard coal-mining code, not especially that it would immediately be followed by all the states, but that it should be a guide and an incentive to each state at those frequent periods when those most interested are trying to improve or otherwise alter their state coal-mining code.

J. I. Thomas, Johnstown, Pa.—It would be difficult to reconcile the differences in the various mining codes and to get all the states to agree to one general code. The Pennsylvania code was adopted in 1911, and there have been but few amendments to it since then, but any effort to bring the

code up to modern standards is seriously opposed by the operators and the miners who do not desire to reopen, as a whole, the 1911 code.

J. J. RUTLEDGE,* Baltimore, Md.—It was my great good fortune some years ago to be a member of the Commission which prepared a code of regulations for the safe operation of metal mines in the State of Wisconsin. It is well to supplement the statute law with such regulations, these regulations to take care of new developments in practice. A quantitative method should be employed to determine whether a mine is gaseous or not. I had nothing to do with the preparation of the Maryland mining laws, but I think they are among the best laws.

G. S. RICE, Washington, D. C. (written discussion).—In this day when there is a constant shifting of the mine personnel from state to state, and when the coal industry is practically all inter-state so far as the sale and transportation of its product is concerned, the variations in mining codes are most unfortunate.

The first is from a safety standpoint. When mine officials move from states where the legal requirements are lax into states with better requirements, they are apt to introduce the dangerous practices to which they have been accustomed. Positive circulation of good air, all mining men will agree, is so vital that without it extensive coal mining could not be carried on. There are no geological boundaries that affect the presence of gases in coal mines. The same kind of coal-mine gases, CH_4, CO_2, N and O normally are found in the anthracite district of Pennsylvania as occur in all the coal-mining districts of the country to the Pacific coast. The variations in quantity of gases found differ from mine to mine within any one district or state and the variations are as much as between the average conditions of any two states.

The other reason is that the coal-mining industry is not restricted by state boundaries in the transportation of and sale of coal, and competitive conditions make uniformity in the cost of safety requirements desirable.

The problem is how can such uniformity be brought about. Federal legislation would be in conflict with state rights. Perhaps one way of approaching this question might be for the coal-mining industry in each state to urge its respective legislature to arrange for the appointment of a technical mining committee which might confer with similar committees from other states with the view to ultimate enactment by each legislature of a uniform code.

When Congress, on Feb. 25, 1920, enacted a law providing for the leasing of all coal land on the Public Domain instead of selling the same outright, it included a provision that before leases were made, the Secretary of the Interior should prescribe regulations governing safety, health, and proper mining of the coal, but which should not conflict with respective state laws. The task of doing this was intrusted to the

* Chief mining engineer, Maryland Bureau of Mines.

Bureau of Mines. The Bureau tried to prepare regulations that would dovetail into the codes of the various states, such as Colorado, Utah, Wyoming, Washington, Montana and other Western states; but this was practically impossible. So a separate code was formed, which provided that where its requirement covering leases in any state were in conflict with the respective state law, the latter as provided in the act would control; however, that if the provisions were in addition to the state requirements then such provisions were applicable.

This statement is made because the Federal lease operating-regulations are included in Mr. Garcia's digest of laws concerning coal mine ventilation and to point out that the bare figures do not give the spirit of the regulations. For example, the operating-regulations require a minimum of 100 cu. ft. of air per man per minute, but that quantity is to be measured at the last crosscut of any split of air. Most state regulations are satisfied with measuring air at the foot of the intake shaft, but investigations have shown that rarely more than one-half of the intake air reaches the faces and, certain records obtained in Illinois showed that less than one-fourth of the air reached the faces. The foregoing and other requirements as regards the quality of the air perhaps calls for furnishing more air to the men at the face than the mining laws of any of the states.

The operating regulations also place the determination of a gassy mine on an analytical basis, the first use of this method in any country except Great Britain. These regulations in their final form were approved by a committee of mine operators and inspectors appointed by the governors of the states in which there was Federal coal land. Since the adoption of the Operating Regulations the Bureau of Mines has given consideration from the experience gained to improvements making for safety, as special problems having arisen through the introduction of new kinds of mining machinery and methods of illumination.

The supervision of the Federal coal leases passed to the U. S. Geological Survey when the Bureau of Mines was transferred (July 1, 1925) from the Interior Department to the Department of Commerce, therefore any views of the Bureau for betterment of coal-mining regulations are now simply recommendatory to the coal-mining industry.

The most important problem in connection with coal-mine ventilation that has been considered by the Mine Safety Board of the Bureau relates to defining a "gassy" or a "non-gassy" mine. The more extensive use of electricity, the introduction of new mining machinery, new methods of illumination and new methods of mining, and the increasing amount of methane encountered in deeper coal-mining makes the question of when or where to use, or not to use, methods, machinery, or appliances that may cause ignition of fire damp, most important.

With the authority given by the Director of the Bureau, I present for your consideration the decision of the Mine Safety Board, classifying the

coal mines on the basis of gas found in them (the word "gas," as it is usually understood in coal mining being synonymous with methane) and also describing a definite method of determining precisely the respective classes.

Classification for Coal Mines

The (U. S.) Bureau of Mines believes that all coal mines are potentially gassy; but for purposes of administration in respect to prevention of explosions and fires, the Bureau recommends the following classification:

Class 1 Coal Mine.—a practically non-gassy mine in which inflammable gas in excess of 0.05 per cent. can not be found by systematic search.

Class 2 Coal Mine.—a slightly gassy mine in which

- (a) inflammable gas has been found,* but in amounts less than 2 per cent. in still air in any active or unsealed-abandoned workings; or
- (b) inflammable gas can be found, but in amount less than 4 per cent. in some place from which the ventilating current has been shut off for a period of 1 hr.; or
- (c) inflammable gas can be found† but in amounts less than ¼ per cent., in a split‡ of the ventilating current; or
- (d) inflammable gas enters a split‡ of ventilating current at a rate§ of not more than 25 cu. ft. per min.

Class 3 Coal Mines.—a gassy mine in which inflammable gas is found in amount greater than specified for a Class 2 Coal Mine.

General Notes Regarding Decision No. 3

(a) The inflammable gas found in coal mines is, with rare exceptions, methane. In coal mining fields where natural gas is found in lower geological horizons by deep wells that pass through or nearby the coal mines, there have been rare instances of a leakage from the well. Natural gas is chiefly methane, almost always more than 85 per cent. is methane, but it usually contains ethane, propane and traces of butane. Therefore, if the latter gases are found in mine air, it is an indication of leakage. The lower limit of explosibility of methane-air mixtures with about 10 per cent. ethane and associated hydrocarbon gases is 4.6 per cent. The limit therefore varies with the character of mixture.

(b) To determine the proper classification of a coal mine, it is advisable that systematic testing and sampling be done at least three times in a period not less than 72 hr. All tests and samples of the mine air, except one, must show an inflammable-gas content less than the maximum limit of the class to which the mine is assigned. In other words, a tolerance of one test or analysis may be permitted to provide for a mistake or a very exceptional occurrence.

(c) When a new mine is being opened in a coal field where existing mines are generally gassy, it is common sense to assume that similar conditions will be found in the new mine, and its development and equipment should be based upon the expectation that it will be assigned to Class 3.

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* By using an approved flame safety lamp, with flame drawn low, or by using an approved gas detector, or by sampling and analysis with an approved gas analytical apparatus.

† By sampling and analysis with an approved gas analytical apparatus, or by using an approved gas detector.

‡ If but one continuous ventilating current is used in a mine this shall be considered a "split" for the purpose of this definition.

§ Determined by sampling, analysis and ventilating-current measurement.

Report of Coal and Coke Committee, American Institute of Mining and Metallurgical Engineers*

(New York Meeting, February, 1926)

During the past year the Committee on Coal and Coke has been collecting data concerning various phases of the bituminous industry about which considerable misinformation has been circulated even, in some cases, through the coal trade papers. The data here presented have been compiled by various members, from the most authentic sources available and, where possible, cover the period 1913–1924. They will enable comparisons to be made within the industry, and of this and other industries, and will explain some of the differences between conditions existing now and in the prewar period.

Capacity and Over-development

The capacity of bituminous mines is frequently stated at figures ranging between 800,000,000 to 1,400,000,000 tons per year. Table 1 shows statistics for the industry, mainly from U. S. Geological Survey sources but partly from U. S. Coal Commission reports. Column 3 gives the mine capacity with equipment and labor as of given date, as estimated from the data published by the U. S. Geological Survey. This is calculated for each State by obtaining the total man days worked as reported by the operators (multiplying average men employed by total days worked at each mine) and dividing this figure into the total production, getting the production per man day; this figure times number of men employed times 308 (a theoretical full working year) equals estimated full-time capacity. Two major objections are made to this method—one that the total number of men employed is used, instead of the number actually working, and the other that the number of days used only deducts Sundays and five holidays. The first objection is answered by the fact that the same number of men is used in estimating capacity as is used in actual production, and while this number is always from 12 to 20 per cent. more than are actually working, this condition exists at all times, although it is worse at times of maximum output. Few coal men will agree with the conclusion that the use of 308 days is correct, as it is practically impossible to keep any mine working full time at the same daily rate of production as when it is working part time, and the opinion is

* Issued Nov. 20, 1925.

general that 280 days is as great a figure as should be used.[1] The capacity as estimated on this basis is given in column 4. This estimate, we believe, is much more nearly the true capacity, although even with present railroad efficiency this amount of coal could not be moved. See Chart 1.

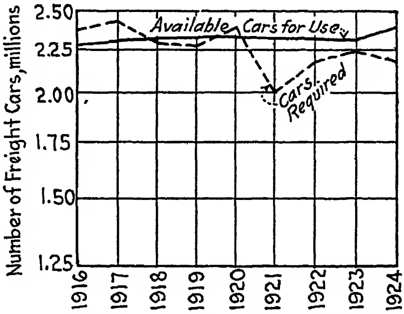

Chart No. 1.—Total number of freight cars available and number required in United States, 1916 to 1924, incl. Data from Interstate Commerce Commission.

In comparing capacity with production, on account of lack of storage facilities, the month of largest production should be used rather than the annual production. These figures for each year are in column 8.

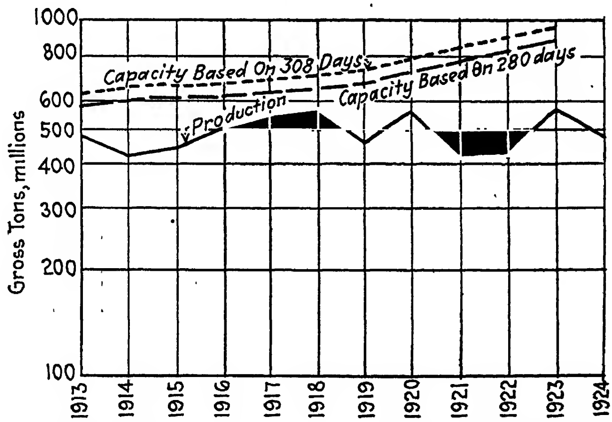

Chart No. 2.—Capacity and production of bituminous coal mines in United States, 1913 to 1924, incl.

Number of men employed (column 5), with exception of 1915, 1916 and 1924, shows a steady increase each year regardless of production.

[1] It is a well-known fact that anthracite mines operate much more steadily than bituminous ones: the greatest number of days worked was 293 in 1918, and the U. S. Coal Commission report says that in recent years anthracite mines have been running 280 to 285 days.

Columns 9 to 12 show all data available about the number of mines.

To determine the truth of the assertion, frequently made, that the soft coal industry is the most over-developed industry in the country, the

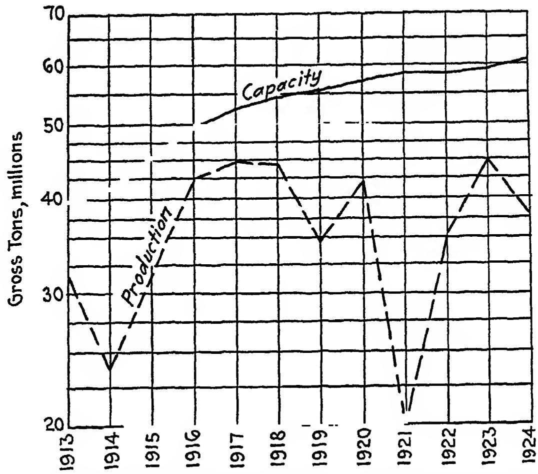

Chart No. 3.—Capacity and production of steel ingots and castings in United States, 1913 to 1924, incl. Data from American Iron and Steel Institute.

data in columns 2, 3 and 4 of Table 1 are shown plotted on Chart 2. In addition the data in Table 2, about the steel industry, and in Table 3, about the copper industry, were secured, and plotted on Charts 3 and 4,

Chart No. 4.—Capacity and production of refined copper in United States, 1913 to 1924, incl. Data from American Bureau of Metal Statistics.

the sources of information being shown. These three industries are the largest mining and metallurgical industries and the comparison of ratios

of production to capacity are shown in Chart 5. This shows that using the ratio of annual production to U. S. Geological Survey estimated capacity, certainly a maximum figure, that the coal industry is less over-developed than copper and only slightly more than steel, on a basis of the theoretical 308-day year and is less over-developed than either the copper or the steel industry on a basis of the more practical 280-day year. Using the ratio of annual production to Coal and Coke Committee estimated capacity shows that the coal industry is less over-developed than either of the other two, and if we use the ratio of largest

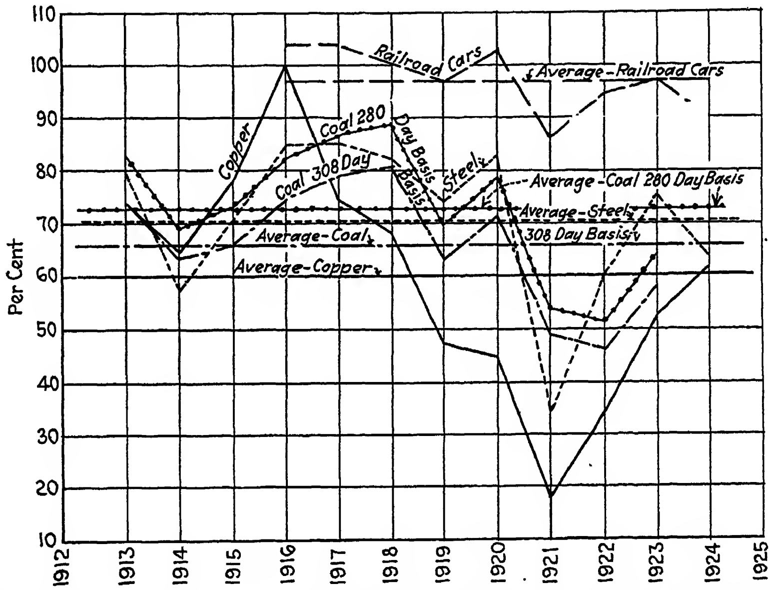

Chart No. 5.—Production in percentage of capacity of railroad cars, steel, coal and copper in United States, 1913 to 1924, incl.

monthly production to either estimated capacity, that both steel and copper are much more over-developed. In considering this last comparison it should be remembered that the products of both steel and copper industries are easily stored.

Growth of Some Items of Costs

Taxation

It has been difficult to secure data showing the increase of taxes since 1913. The only state having this data available directly is New Mexico, where annual assessments are made for the various industries and are based on the earning power of each one. Table 5 shows the taxes per acre

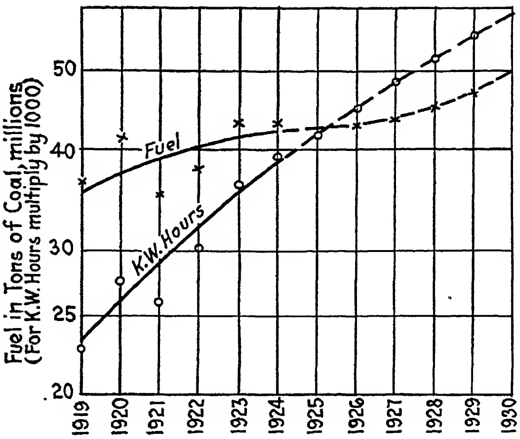

CHART NO. 6.—TOTAL KW. HRS. AND FUEL USED BY PUBLIC UTILITIES IN UNITED STATES. DATA FOR 1919–1924 FROM U. S. GEOLOGICAL SURVEY; ESTIMATED TO 1929.

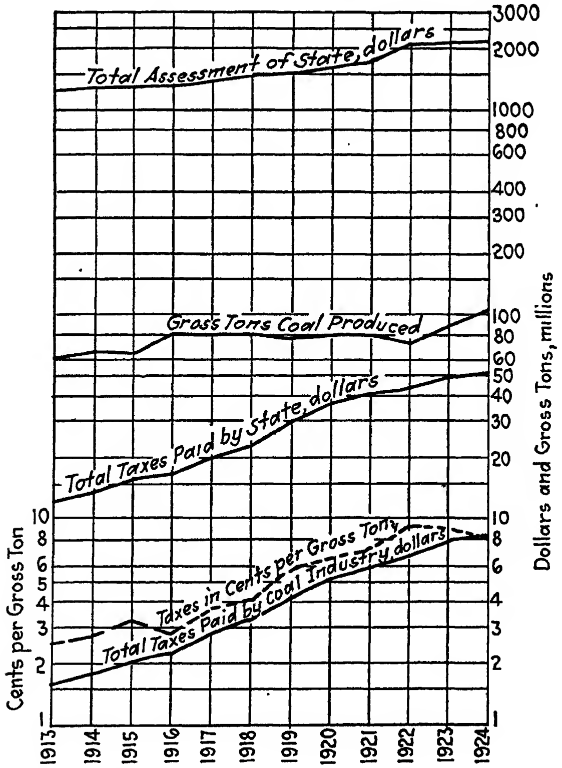

CHART NO. 7.—TAXATION AS FACTOR IN COAL INDUSTRY OF WEST VIRGINIA. ALL TAX DATA FROM STATE TAX COMMISSION; PRODUCTION FROM STATE DEPARTMENT OF MINES. PER CENT. OF TAXES PAID BY COAL INDUSTRY IN 1917 AND AFTERWARDS IS SAME AS SHOWN BY STATE TAX COMMISSION REPORT FOR THAT YEAR. DATA COMPILED FOR AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS BY COMMITTEE ON COAL AND COKE.

and per ton for this state. The increase here has been much less than in most other coal-producing states, largely owing to the depressed condition of the coal industry due to oil competition.

In West Virginia all assessments and taxes paid are reported to the State Tax Commissioner and until 1917 the taxes paid by various industries were segregated. By combining the ratio of the taxes on coal with the total for that year with the figures for later years, it has been possible to prepare Table 6, and it is believed that these figures for coal are less, rather than more, than the actual amounts paid. The total taxes per ton produced show a steady increase from 2.5 cts. in 1913 to 9.2 cts. in 1922; the reduction in the two later years is due to the large increase in production and not to any reduction in taxes. Data are shown in Chart 7.

No other states have such data available. The information in Table 8 for Pennsylvania was obtained from many of the larger companies for each county, and while not complete, is thought to be representative.

For Illinois, the only data available[1a] are:

Year	Per Ton	Year	Per Ton
1914	$0.0015	1921	$0.015
1917	0.005	1922	0.027
1919	0.010	1923	0.025
1920	0.016	1924	0.025

Supplies

The only general data for the industry available are:

	1916[a]	1917[a]	1918[a]	1920[b]	1921[c]
Cost per ton	$0.107	$0.179	$0.259	$0.268	$0.370

[a] Brief of Bit. Opers. Special Comm. to U. S. Coal Commission (July 30, 1923). Data compiled from Federal Trade Comm. report, *Coal* (June 30, 1919).

[b] Brief of Bit. Opers. Special Comm. to U. S. Coal Commission (July 30, 1923). Data compiled from Federal Trade Comm. report (1922).

[c] Statement of J. D. A. Morrow, Vice Pres., National Coal Ass'n. (April 25, 1922). I. C. C. Docket 13,293.

Illinois[2]

Coal Year Apr. 1–Mar. 31	Cost per Ton	Coal Year Apr. 1–Mar. 31	Cost per Ton
1914–15	$0.106	1921	$0.449
1916	0.120	1922	0.349
1917	0.134	1923	0.348
1918	0.248	1924	0.271
1919	0.322	1925	0.231
1920	0.323		

[1a] Supplied by Honnold Coal Bureau. None of above figures include Federal taxes.

[2] Supplied by Honnold Coal Bureau.

Power and Fuel for Public Utilities

The data shown in Table 4 are taken from information published by the U. S. Geological Survey and from which the curves in Chart 6 are plotted, indicating the growth in power produced and fuel required by public utilities from 1919 to 1924, and an estimate of the growth from 1924 to 1929, inclusive. These curves clearly indicate the great increase in efficiency in the use of fuel by the public utilities. They also show the very rapid increase in the use of power, and indicate that the increase in efficiency has been very rapid up to this time and that the increase in efficiency in the near future probably will not be as great as it has been in the near past. This means that the increase in fuel will more nearly follow the increase in production of power.

Fuel for Railroads

The fuel for locomotives on the railroads of the United States consists of bituminous coal, anthracite coal and fuel oil. Of these fuels bitumi-

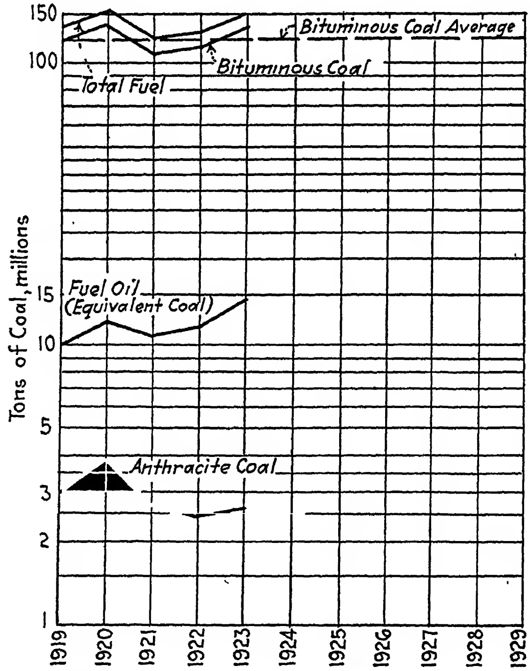

Chart No. 8.—Total railroad fuel consumed in United States, 1919 to 1923, incl. Data for 1919–1923 from Interstate Commerce Commission; estimated to 1929.

nous coal represents about 90 per cent., anthracite 2 per cent., and fuel oil 8 per cent. of the total. The amount of wood used is only a small fraction of 1 per cent.

From information obtained from the Interstate Commerce Commission the fuel requirements for railroads for 1919 to 1924 inclusive, are given in Table 7 and plotted in Chart 8. The anthracite curve indicates a slightly reduced consumption during the period while the fuel oil shows a fairly definite increase. This increase will probably keep up until 1925–1926, when a maximum will be reached after which it is probable that a definite decrease will start due to the greater demands for gasoline and lubricating oils, and the smaller amount of fuel oil available. In fact, some of the railroads in the West are now contemplating changing certain divisions back to coal, while some of the metallurgical industries are changing to powdered coal.

The curves on Chart 8, showing the consumption of bituminous coal and total fuel, are practically parallel and indicate an almost constant demand over the period.

The increase in freight traffic of the railroads of the United States each year is approximately 6 per cent. The electrification of railroads is proceeding very slowly from a mileage standpoint, but considerably faster from a tonnage standpoint, as the most congested sections are the first to be electrified. Some of this electric power will be generated in hydroelectric plants and that portion generated in steam plants will use fuel much more economically than will a steam locomotive.

A number of economies are being introduced in the design and construction of steam locomotives and also in the manner of handling the fuel. New locomotives will average of larger size than those replaced and these new locomotives will use fuel more economically.

A still further economy is expected to result from running freight trains in larger units and at higher speeds. Some roads are contemplating running freight trains on regular high-speed schedules with passenger trains.

For the above reasons it is probable that the demand for coal for railroad fuel during the next 10 years will remain about stationary, notwithstanding the increase in traffic of approximately 6 per cent. each year.

Past and Future Fuel Production in United States

The four principal fuels used in the United States, and the percentage of each at this time, are:

Bituminous coal	73 per cent.
Anthracite	11.5 per cent.
Fuel oil	10.3 per cent.
Natural gas	5.2 per cent.

Although coal always has been the principal fuel, natural gas started to make itself felt in the 90's and fuel about 1905.

From information obtained from the American Petroleum Institute, U. S. Geological Survey, American Institute of Mining and Metallurgical Engineers, Standard Statistics Co., and Mining and Metallurgical Society of America, the information in Chart 9 has been plotted showing the production of natural gas and fuel oil in this country from 1904 to 1924. The curves have been extended to 1940, these extensions being based on gradual failure of the natural gas supply and changing back to coal as a fuel by many industries due to increase in cost and reduced supply of fuel oil on account of the future great demands for gasoline and lubricating oils.

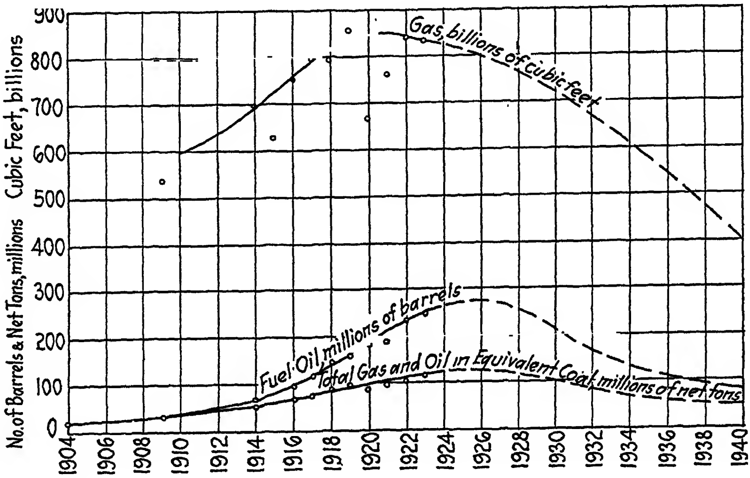

CHART No. 9.—PAST AND ESTIMATED FUTURE OIL AND GAS PRODUCTION OF UNITED STATES AND TOTAL EQUIVALENT IN COAL. SOURCES OF DATA: AMERICAN PETROLEUM INSTITUTE, AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS, STANDARD STATISTICS CO., U. S. GEOLOGICAL SURVEY AND MINING AND METALLURGICAL SOCIETY OF AMERICA.

Chart 9 also shows a curve indicating the equivalent coal which would be needed to supply the fuel when gas and fuel oil are used. These curves indicate that the peak in use of gas and fuel oil is about reached and a reduction in the use of these fuels should be expected in the future.

Chart 10 shows the production of fuel in the United States from 1860 to 1924, with the curves extended to 1950. The anthracite and bituminous coal values up to 1924 were obtained from the U. S. Geological Survey and the total fuel curve is made up from the coal production with the information on Chart 9 added to take care of the oil and gas. The total fuel curve was extended to 1950 by extending a fuel per capita curve and using Chart 11 to obtain the expected population. These extensions are not intended as a prophecy but the values used are based on the best data obtainable at this time.

The future production of anthracite indicates only a small increase to a maximum in 1935. The bituminous curve is estimated by subtract-

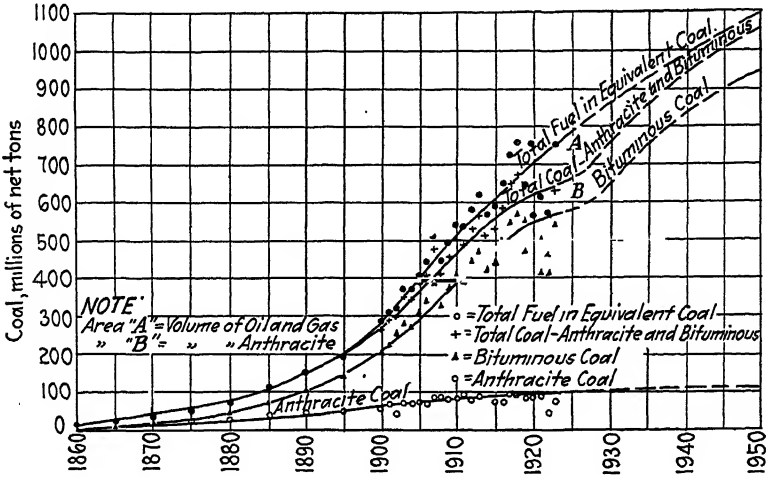

CHART NO. 10.—PAST AND FUTURE FUEL PRODUCTION IN UNITED STATES; TOTAL FUEL; TOTAL COAL—BITUMINOUS AND ANTHRACITE; TOTAL BITUMINOUS COAL; TOTAL ANTHRACITE. DATA FROM U. S. GEOLOGICAL SURVEY, AMERICAN PETROLEUM INSTITUTE, MINING AND METALLURGICAL SOCIETY OF AMERICA AND AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS.

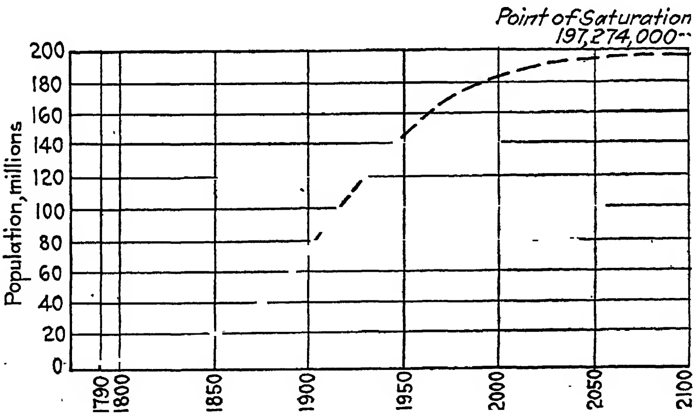

CHART NO. 11.—GROWTH OF POPULATION IN UNITED STATES, FROM PEARL'S "BIOLOGY OF DEATH," p. 250. POINT OF SATURATION AS INDICATED IN CHART IS 197,274,000 POPULATION IN 2100. VALUES FROM 1790 TO 1920, INCL., FROM BUREAU OF THE CENSUS.

ing the anthracite, as shown on the anthracite curve, and the fuel oil and gas, as shown on Chart 9, from the total fuel.

The bituminous curve has a very decided kink between 1920 and 1930, caused by the use of fuel oil and gas. The extension of the curve shows,

however, that the lost ground is being gradually regained due to the falling off of the natural gas supply and the confining of fuel oil principally to marine service and a restricted land service.

TABLE 1.—*Bituminous Coal Industry Statistics (Net Tons)*

Year	Production	U. S. Geol. Survey Estimated Mine Capacity 308 Days—Present Equipment	Coal and Coke Comm. Estimated Mine Capacity 280 Days—Present Equipment	Men Employed	Average Days Worked
1	2	3	4	5	6
1913	478,435,297	635,164,000	577,422,000	571,882	232
1914	422,703,970	667,656,000	606,960,000	583,506	195
1915	442,624,426	671,568,000	610,516,000	557,456	203
1916	502,519,682	672,939,000	611,763,000	561,102	230
1917	551,790,563	699,389,000	635,808,000	603,143	243
1918	579,385,820	716,670,000	651,518,000	615,305	249
1919	465,860,000	735,820,000	668,927,000	621,998	195
1920	568,667,000	796,134,000	723,758,000	639,547	220
1921	415,921,950	859,758,000	781,598,000	663,754	149
1922	422,268,099	915,905,000	832,641,000	687,958	142
1923	564,156,917	970,728,000	882,480,000	702,817	179
1924	483,280,000	870,000,000	791,000,000	619,604	171

Year	Average Price Realized	Production of Largest Month	No. of Mines (Excluding Wagon Mines)	Mines Opened (b)	Mines Abandoned (b)	Wagon Mines Shipping by Rail
	7	8	9	10	11	12
1913	$1.18	46,164,000	5,776	....	....	(a)
1914	1.17	45,455,000	5,592	....	....	(a)
1915	1.13	45,814,000	.....	....	....	(a)
1916	1.32	46,593,000	.....	454	201	(a)
1917	2.26	48,337,000	6,939	1285	207	(a)
1918	2.58	55,114,000	8,319	1573	458	(a)
1919	2.49	57,200,000	8,994	1058	253	(a)
1920	3.75	53,278,000	8,921	401	419	4,405
1921	2.89	44,687,000	8,038	511	....	(a)
1922	3.02	51,936,000	9,299	....	....	4,851
1923	2.68	51,903,000	9,238	....	....	2,384
1924	2.20	52,464,000	7,585	....	....	(a)

(*a*) Canvass not made.
(*b*) Information not available for other years than those shown.
Data from U. S. Geological Survey.

TABLE 2.—*Annual Production and Capacity of Steel Ingots and Castings, Dec.* 31, 1913–1924[3]

Year	Production, Total Gross Tons	Capacity, Total Gross Tons	Year	Production, Total Gross Tons	Capacity, Total Gross Tons
1913	31,300,874	39,689,265	1919	34,671,232	55,637,135
1914	23,513,030	41,293,880	1920	42,132,934	57,376,810
1915	32,151,036	45,787,780	1921	19,783,797	58,416,680
1916	42,773,680	49,613,888	1922	35,602,926	58,644,655
1917	45,060,607	52,541,445	1923	44,943,696	59,431,710
1918	44,462,432	54,482,740	1924	37,931,939	61,136,805

[3] Data from American Iron and Steel Institute.

TABLE 3.—*Annual Production and Capacity of Refined Copper in United States, 1913–1924, Inclusive*[4]

Year	Production in Pounds	Capacity in Pounds	Year	Production in Pounds	Capacity in Pounds
1913	1,224,484,098	1,768,000,000	1919	1,286,419,329	2,746,000,000
1914	1,150,137,192	1,778,000,000	1920	1,209,061,040	2,702,000,000
1915	1,388,009,527	1,892,000,000	1921	505,586,098	2,718,000,000
1916	1,927,850,548	2,496,000,000	1922	950,285,947	2,718,000,000
1917	1,886,120,721	2,794,000,000	1923	1,434,999,962	2,718,000,000
1918	1,908,533,595	2,794,000,000	1924	1,628,000,000	2,692,000,000

TABLE 4.—*Total Power and Fuel Used in U. S. A. by Public Utilities, 1919–1924; Estimated to* 1929[5]

	Kw.-hrs. Produced 1,000,000	Pounds of Coal per Kw.-hr.	Total Fuel in Tons of Coal 1,000
1919	22,775	3.2	36,500
1920	27,600	3	41,420
1921	26,000	2.7	35,240
1922	30,400	2.5	38,000
1923	36,200	2.4	43,522
1924	39,200	2.2	43,130
1925	41,900	2	41,900
1926	45,000	1.9	42,800
1927	48,500	1.8	43,750
1928	51,900	1.75	45,400
1929	55,300	1.7	47,000

TABLE 5.—*Taxes Paid in New Mexico*, 1918–1924[6]

Year	Taxes Paid	Acres Assessed	Taxes per Acre	Taxes per Ton
1918	$110,292.77	256,000	$0.43	$0.037
1919	145,590.73	256,000	.57	.062
1920	207,340.18	256,000	.81	.075
1921	201,694.60	256,000	.79	.110
1922	195,277.63	256,000	.76	.083
1923	176,900.28	256,000	.69	.081
1924	209,074.63	256,000	.82	.109

[4] Data from American Bureau of Metal Statistics.

[5] Data from U. S. Geol. Surv. to 1924. Estimated to 1929 by R. N. Davis, Bonbright prize award.

[6] Figures include approximately 75 per cent. of output of state.

TABLE 6.—*Growth of Taxes in West Virginia*, 1913–1924[7]

Year	Total Assessment of State	Total Taxes Paid by State	Total Taxes Paid by Coal Industry[8]	Gross Tons Coal Produced	Taxes in Cents per Gross Ton
1913	$1,243,315,543	$12,146,299	$1,566,873	61,770,352	2.5
1914	1,282,438,578	13,698,448	1,767,100	65,783,088	2.7
1915	1,286,569,524	15,871,577	2,047,433	64,118,677	3.2
1916	1,298,550,852	16,584,637	2,238,926	79,612,298	2.8
1917	1,376,139,828	20,265,456	2,857,429	79,806,652	3.6
1918	1,449,451,754	22,471,421	3,168,470	81,041,640	3.9
1919	1,490,773,132	29,853,845	4,209,392	75,875,493	5.5
1920	1,579,594,399	36,339,460	5,123,864	79,991,316	6.4
1921	1,696,068,361	40,692,708	5,737,671	80,761,604	7.1
1922	2,092,556,969	42,968,374	6,502,453	70,888,203	9.2
1923	2,109,813,186	48,868,693	7,822,048	87,031,408	9.0
1924	2,122,919,846	51,512,317	8,202,796	103,325,960	7.9

In 1915[9] tax paid by coal was 12.9 per cent. of entire state.
In 1916 tax paid by coal was 13.5 per cent. of entire state.
In 1917 tax paid by coal was 14.1 per cent. of entire state.

TABLE 7.—*Fuel Requirements for Railroads in United States*, 1919–1923, *Inc.*[10]

Year	Total Railroad Fuel, U. S.		Fuel Oil (Gal.)	Fuel Oil Equivalent Tons of Coal	Total Fuel (Tons)
	Anthracite Coal	Bituminous Coal			
1919	2,981,959	119,692,067	1,586,061,174	9,900,000	132,620,935
1920	3,860,970	135,413,695	1,929,670,624	12,020,000	151,405,712
1921	2,643,724	107,910,146	1,661,443,618	10,750,000	121,006,242
1922	2,472,652	113,163,083	1,828,125,050	11,400,000	127,213,343
1923	2,614,576	131,491,561	2,334,365,782	14,600,000	147,921,714
1924	2,500,000[11]	115,000,000	2,520,000,000	15,700,000	133,200,000

[7] All tax data from reports of State Tax Commissioner.

[8] Total tax paid by coal is obtained by adding to the gross sales tax from coal 14.1 per cent. of all other taxes paid. Assessments of coal-producing counties show that this figure, obtained for 1917, is conservative.

[9] In 1915, 1916 and 1917 taxes paid by industries were segregated. After 1917 no such record is available.

[10] Data from Interstate Commerce Commission.

[11] Estimated.

TABLE 8.—*Taxes Paid by Bituminous Coal Industry, Pennsylvania, 1914–1924, Inclusive*

Allegheny County				Center County			
Year	Taxes Paid	Acres Assessed	Taxes per Acre	Year	Taxes Paid	Acres Assessed	Taxes per Acre
1914	$103,171	31,214	$ 3.30	1914	$ 850	8,182	$ 0.10
1915	100,435	29,756	3.37	1915	850	8,182	0.10
1916	175,094	32,630	5.37	1916	867	8,182	0.11
1917	208,917	40,273	5.19	1917	868	8,182	0.11
1918	229,143	39,383	5.82	1918	826	8,182	0.10
1919	329,321	37,558	8.77	1919	886	8,182	0.11
1920	401,280	36,701	10.93	1920	1,169	8,182	0.14
1921	456,494	38,846	11.75	1921	1,179	8,182	0.14
1922	471,880	37,891	12.45	1922	1,190	8,182	0.14
1923	460,082	37,069	12.41	1923	1,190	8,182	0.14
1924	491,178	36,048	13.63	1924	1,229	8,182	0.15
1925	510,308	35,330	14.44				
Armstrong County				Clearfield County			
1914	8,761	17,933	0.48	1914	5,306	27,963	0.19
1915	8,766	18,395	0.47	1915	5,451	27,963	0.19
1916	9,493	18,392	0.51	1916	6,443	27,279	0.24
1917	11,034	22,392	0.49	1917	6,530	27,279	0.24
1918	16,145	25,343	0.63	1918	7,065	27,279	0.26
1919	28,191	25,482	1.14	1919	8,053	27,279	0.29
1920	30,805	25,514	1.20	1920	8,571	27,279	0.31
1921	31,239	25,535	1.22	1921	9,566	27,279	0.35
1922	48,281	27,481	1.75	1922	10,655	27,279	0.39
1923	42,184	27,506	1.53	1923	12,100	27,279	0.44
1924	44,527	27,703	1.60	1924	13,806	27,279	0.51
Cambria County				Fayette County			
1914	29,412	63,475	0.46	1914	551,462	41,346	13.34
1915	29,282	63,475	0.46	1915	558,500	40,604	13.75
1916	38,807	63,475	0.61	1916	629,329	40,481	15.54
1917	38,539	60,575	0.63	1917	688,497	39,457	17.45
1918	48,252	60,575	0.80	1918	756,765	38,342	19.74
1919	52,222	60,575	0.86	1919	928,567	37,126	25.01
1920	70,649	61,475	1.15	1920	1,191,218	35,851	33.23
1921	62,269	57,050	1.09	1921	1,376,413	38,484	35.76
1922	68,294	57,050	1.20	1922	1,765,133	38,657	45.66
1923	76,898	55,715	1.38	1923	1,746,885	38,240	45.68
1924	73,884	55,786	1.32	1924	1,747,763	37,515	46.59

TABLE 8.—(*Concluded*)

Greene County				Somerset County			
Year	Taxes Paid	Acres Assessed	Taxes per Acre	Year	Taxes Paid	Acres Assessed	Taxes per Acre
1914	$ 4,676	5,386	$0.87	1914	$ 25,535	53,744	$ 0.47
1915	7,258	5,399	1.34	1915	26,031	53,744	0.48
1916	5,900	5,687	1.04	1916	27,501	53,733	0.51
1917	7,328	5,687	1.29	1917	32,283	53,932	0.60
1918	8,540	5,687	1.50	1918	33,764	53,999	0.62
1919	15,824	5,681	2.78	1919	37,483	54,925	0.68
1920	21,373	5,667	3.77	1920	49,060	54,859	0.89
1921	26,520	8,064	3.29	1921	86,433	64,635	1.34
1922	36,061	8,714	4.14	1922	86,346	69,778	1.24
1923	33,278	8,697	3.83	1923	80,534	69,743	1.15
1924	52,752	8,535	6.18	1924	78,078	69,796	1.12

Indiana County				Washington County			
1914	12,696	28,507	0.45	1914	184,148	91,445	2.01
1915	12,672	28,507	0.45	1915	186,523	90,664	2.06
1916	17,181	30,581	0.56	1916	213,949	98,494	2.17
1917	13,436	26,201	0.51	1917	289,832	101,823	2.85
1918	16,405	26,201	0.62	1918	310,279	108,746	2.85
1919	22,086	26,201	0.84	1919	364,400	114,010	3.20
1920	25,460	26,201	0.97	1920	474,999	114,044	4.16
1921	28,215	26,008	1.08	1921	568,364	125,264	4.54
1922	32,912	26,008	1.27	1922	593,732	126,061	4.71
1923	37,565	26,760	1.40	1923	620,970	125,433	4.95
1924	44,968	27,100	1.66	1924	613,457	125,802	4.88
				1925	582,991	119,856	4.86

Jefferson County				Westmoreland County			
1914	3,161	4,056	0.77	1914	345,693	40,259	8.59
1915	2,826	4,169	0.67	1915	335,356	40,323	8.32
1916	2,751	4,127	0.66	1916	388,096	38,507	10.08
1917	2,589	4,200	0.63	1917	395,830	37,081	10.67
1918	2,804	4,200	0.66	1918	446,586	35,914	12.43
1919	3,358	4,203	0.79	1919	504,962	36,610	13.79
1920	3,688	4,183	0.88	1920	646,884	35,622	18.16
1921	4,535	4,203	1.07	1921	688,021	38,438	17.90
1922	2,291	4,202	0.54	1922	719,546	35,757	20.12
1923	2,459	4,209	0.58	1923	722,428	35,374	20.42
1924	2,563	4,218	0.60	1924	746,655	34,525	21.63

TABLE 9.—*Growth of Coal Industry Taxes, Ohio,* 1916–1924 *Incl.*[12]

Year	Total Assessment of Coal Property[13]	Taxes Paid	Net Tons Coal Produced	Taxes in Cents per Net Ton
1916	$21,763,760	$ 318,186	33,993,399	0.94
1917[14]	30,636,770	450,360	40,441,819	1.11
1918	38,665,400	587,626	45,180,278	1.30
1919	42,416,800	708,360	33,377,463	2.12
1920	43,795,510	845,251	40,568,304	2.08
1921	46,153,500	946,145	30,782,765	3.07
1922	45,292,090	978,307	24,405,382	4.01
1923	46,075,840	1,013,667	37,959,589	2.67
1924	44,778,530	984,126	27,663,819	3.56

[12] Data from Pittsburgh Vein Operators' Assoc. of Ohio. Figures cover 20 counties in Ohio reporting coal mining; production and taxes paid by stripping operations not included.

[13] Coal mined under royalty agreement not included in "Total Assessment of Coal Property."

[14] Ohio Tax Commission increased tax valuation of coal from 33⅓ to 50 per cent.

DISCUSSION

J. A. GARCIA, Chicago, Ill.—Is the increase in capacity due to increased facilities of existing operation, or to the construction of new mines? By capacity does the committee mean actual production, hoisting capacity of the shaft, supply of railroad cars, or something else?

H. N. EAVENSON, Pittsburgh, Pa.—Mine production statistics show that over 80 per cent. of the mines produce under 1000 tons a day. The number of large mines opened has been very small. Many of the new mines were opened during the war; these were very small and many of them have now been abandoned.

The capacity of the mines as reported is based on the actual production; that is, by taking the number of men actually employed, the number of days worked, and the actual production, and has nothing to do with the actual capacity of any one plant. The mines should actually be able to operate on a 280-day basis and get out that much coal.

J. A. GARCIA.—The coal produced is consumed, but the market is in part regulated by the capacity of the mines and their ability to get out more coal. Capacity is one thing and production is another.

H. N. EAVENSON.—A better way to designate it is as potential or possible production. If the mines had the orders, the men, and the cars, they could produce that much coal.

T. DeVenny, Edgarton, W. Va.—Do the figures in Table 1 indicate that any improvements in mining methods have increased the efficiency of the men employed?

H. N. Eavenson.—The number of men employed has steadily increased, with the exception of the two years 1915 and 1916. On account of the number of days worked, the output per man per day has increased.

G. S. Rice, Washington, D. C.—There are curves that show that the output has risen from an average of 2½ tons per man per day, 30 years ago, to 4½ tons now; this is chiefly due to machine mining.

H. N. Eavenson.—I do not think that the efficiency is as large as we can expect, but in a well-managed plant, the average production per day is far more than these figures. In West Virginia, some plants have been running at almost full capacity and mining more coal than ever; in fact, last year the State produced almost as much as Pennsylvania, yet hundreds of mines in the State were not operated at all.

T. T. Read, New York, N. Y.—If a man works 150 days in a mine this year he is counted as one man. Next year he may work 75 days in one mine and 75 in another, then he is counted as two men; how can you remedy that?

C. E. Lesher, Pittsburgh, Pa.—The figures in the table are based on the average number of men throughout the year. They were carefully checked by the Coal Commission by a study of payrolls, and the individual names were checked through for the year 1921 for a thoroughly representative list of mines, comparing these with the reports made by the operators. The statistics are all based on reports issued by the U. S. Geological Survey. They represent the average number of men on the payroll, which is about all the coal operator can report and it is comparable from year to year.

T. T. Read.—If a man works part of the year in one mine, and part of the year in another would he not appear as two men employed in coal mining?

C. E. Lesher.—Not at the same time.

E. W. Parker, Philadelphia, Pa.—The report of the U. S. Coal Commission gives an erroneous idea as to the number of men employed in the anthracite mines. The statisticians of the Commission included all the names upon the payrolls of every company and did not make any allowance for labor turnover. Consequently, if a man was employed by different companies during the year, or even at different mines of the same company, he appeared as a different man in each employment.

According to the Commission's report, 76,017 miners' laborers were employed in 1921, whereas the largest number employed in any one month was 24,588, and the average for the year was about 22,000. The Commission reported 64,275 contract and consideration miners employed in 1921; but the largest number of miners employed was 44,515 and the average number 43,400.

The annual reports of the U. S. Geological Survey give the average number of men employed during the year. These averages are based on the average numbers of men on the payrolls each payday. The difference in the "average earnings" when 76,000 men are used as the diversor, instead of 22,000, is readily seen.

Evaluation of Coal

[Abstract of remarks by R. H. Sweetser and subsequent discussion at the New York Meeting, February, 1926. The TRANSACTIONS will ultimately contain a report of work resulting from these discussions.]

Need for a standard of valuation.—There should be an accepted method for evaluating the different kinds of bituminous coals based on their carbon just as that for iron ores is based on iron. Variation in the classes of coals and their desirability for the different industrial uses is analagous to the case of iron ores. The iron people have developed a standard. In the past coals have not been intelligently distributed because the public has not been informed as to its uses. Many operators have known nothing about the final disposal of the coal shipped from their mines. Sales agents and retailers have had no means of evaluating coals except to make the price as high as the market will stand. Producers and distributors need to know how much more valuable the coals would be if prepared so as to contain less objectionable material.

Mr. Sweetser called on the Coal and Coke Committee of the Institute to organize a study through producers and users to develop a method scientifically and commercially sound.

E. A. Holbrook said that the information exists but the public is unwilling to investigate. He praised Ashley's[1] nomenclature of the different kind of coal which has been accepted by the Coal Mining Institute of America and recommended to the American Engineering Standards Commitee. G. S. Rice pointed out the wider variety and distribution of consumption of coal compared to iron ore. C. M. Young approved Mr. Sweetser's remarks and hoped for a research institution, probably endowed, to study utilization of coal after it comes from the ground.

Mr. Sweetser cited incidents to prove his former statements. He said that a reduction of 1 per cent. ash in the coal would cut down the cost of pig iron at a blast furnace 30 c. a ton, that the coal men should tell what the iron men would have to pay to get better coal. H. J. Preyn endorsed the need of blast-furnace operators to know the kind of coal from which their coke is made and the latter's ash content, as this affects coke consumption.

A. C. Fieldner explained that the difficulty of evaluating coal on the basis of chemical analyses for calorimetric tests is: (1) representative

[1] G. H. Ashley: A Use Classification of Coal. *Trans.* (1920) **63**, 782.

samples are hard to get and (2) there are no standard tests to determine such important qualities as friability, tendency to slack on standing, and behavior in the fuel bed. He described the methods of the Government in purchasing coal and praised Professor Parr's[2] scheme of classification in which the factors are: (1), the calorific value of moisture-and-ash-free substances and (2), the moisture-and-ash-free volatile matter. He endorsed Mr. Sweetser's proposal as did W. H. Blauvelt who also pointed out that one evaluation scale would be required for making coke for water gas and another for other uses. H. N. Eavenson suggested limiting work for the present to coal for furnaces. Mr. Sweetser declared that steel men must know about coal for many other uses than making coke.

[Abstract of discussion at Pittsburgh Meeting, October, 1926. A. I. M. E., A. S. T. M., American Gas Assn., Eastern States Blast Furnace and Coke Oven Assn., Gas and Fuel Division of American Chemical Society, and Southern Ohio Pig Iron and Coke Assn. were represented.]

Combustibility of coke.—The object was to obtain a common understanding of the term "combustibility." R. H. Sweetser said that blast-furnace operators gaged the combustibility of coke by the rate of flow through the zone of combustion. He held that coke-oven operators can change the combustibility of coke. H. J. Rose said that size and screening greatly affect combustibility of coke. It was generally acknowledged that high combustibility is a desirable property of coke and that size is an important factor of combustibility.

A. R. Maxwell stated that the blast furnaces want coke that has not been burned too hard and has open-cell structure. It must be strong enough to carry the burden. P. Nichols discriminated between "combustibility" and "reactivity." Combustibility means that something burns easily or ignites readily. J. D. Davis thought reactivity, in blast-furnace practice, merely one factor of combustibility although it is sometimes wrongly termed combustibility when used in connection with laboratory tests. Reactivity is rate of chemical action of oxidizing gas with coke. He showed slides of laboratory results. O. P. Hood thought the problem of getting air to the coke surface must be settled before combustibility, as illustrated by Mr. Davis, can be discussed. Mr. Nichols explained his method of test and showed that loss of heat from furnace walls and the heating of the descending coke affect the fuel-bed temperature and could produce a continual fall in spite of chemical reaction having ceased, so that temperature curves cannot be used as a scale for combustibility.

Testing of coke.—The physical testing of coke at present is unsatisfactory because of the difficulty of selecting representative samples and

[2] S. W. Parr: The Classification of Coals. *Jnl.* Am. Chem. Soc. (1906) 28, 1425; *Jnl.* Ind. Eng. Chem. (1909) 1, 636, (1922) 14, 921.

the doubtful practical significance of the results. W. B. Havens said that the Eastern States Blast Furnace and Coke Oven Association and the Chicago Blast Furnace and Coke Oven Association had appointed a committee to try to reduce the results of different investigators to a commonly understood nomenclature. The Eastern States association is agreed that coke cannot be properly classified as to its general value as a blast-furnace fuel by any test, although this has been possible for individual operators. The standard porosity and shatter tests of the A. S. T. M. and its tumbling test under consideration were discussed. Square holes had been judged best for screen tests. The value of the shatter test was confirmed and the need for standard physical tests in order to correlate results was stressed.

Appraisal of Coal-property Values

By H. M. Chance, Philadelphia, Pa.

(Pittsburgh Meeting, October, 1926)

The present value of most coal properties resides largely in the coal remaining to be mined, which thus constitutes the most important asset. The object of this paper is to discuss methods commonly used in computing the value of minable coal.

To distinguish clearly between such coal values and the values of other property, the several kinds of property values should be divided into classes; so as to avoid the unintentional omission of items of importance.

Value of Coal Reserves

Judicial decisions prescribe, and appraisers commonly accept, current sales value as the preferred means for determining the value of transferable property. With few exceptions a property is worth the sum that can be realized from its sale by a willing seller to a willing buyer, a sale in which compulsion urges neither party, and between whom there is no community of interest to fix a value; or it is worth at least as much as a responsible buyer will offer for it. Actual sales of like property or bids from responsible buyers for the property, or for substantially like property, are the best established methods for determining values.

Only in the absence of such sales or bids can valuations by other methods be entertained, and only for reasons of weight can sales values be set aside.

Valuation Based on Royalties

Sales of coal in the ground, as customarily carried out through lease under royalty agreements, are therefore the most trustworthy means available for determining the value of coal reserves.

The constantly rising value of high-grade coal in the United States can most readily be shown by the increasing royalties per ton that can be obtained for such coals under leasehold. The rising value of land containing such coal can also, of course, be proven by the price at which such properties are actually sold as compared with prices obtainable for lands containing coal of the same grade in the past. It is exceedingly difficult, however, to establish values by the prices paid for coal

property, because the deeds or instruments of conveyance rarely disclose the actual price paid for the property, the consideration named being a nominal sum, or a sum much below the price actually paid. Hence an examination of the recorded deeds may fail to establish either present or past values. On the other hand, leasehold agreements always state the royalty per ton. As many, and the most important, of such leasehold agreements are recorded, the enhancement in value of coal, and especially of coals of high grade, becomes a matter of public record.

Coals of mediocre quality, or coals presenting mining conditions adverse to low-cost production, often show small, uncertain and erratic changes in value; for some coals present values are lower than past values, and in many properties of poor grade, it would be difficult to show regular or persistent tendency to increased value.

Throughout the United States, medium and relatively low-grade coals are in abundant supply, whereas the areas containing coal of high grade form a very small percentage of the total coal acreage of the United States.

Characteristics of High-grade Coals

Although high-grade coals can be produced only from circumscribed areas, the demand for high-grade fuels steadily grows—a condition of decreasing available supply and increasing current demand. It is therefore necessary to recognize the generic difference between high-grade coals on the one hand and all other coals on the other.

Bituminous coals entitled to be ranked as of high grade are divisible into three classes:

1. Coals with low volatile, low sulfur, low ash, high B. t. u.; and as a special sub-class, those with high-fusion-point ash.

2. Coking coals with low ash and low sulfur, or which by cleaning can readily be improved to make high-grade coke for blast furnace and foundry use; and as a sub-class coal suitable for by-product coking with relatively high yield of by-products, or available nitrogen (ammonia) or both.

3. Coals finding special markets, especially those in demand for domestic (household) use such as splint coal, hard blocky coal of good grade, cannel coal; and coals especially adapted to meet the requirements of some industrial use or manufacture, for example, to make gas either for manufacturing or domestic use.

Anthracite having physical and chemical properties equal to or approximating the anthracite of Pennsylvania, is a high-grade coal and, like high-grade bituminous coals and for the same reasons, has shown steadily rising value.

Valuation Based on Earnings

For many years there has been a growing tendency to base the valuation of the property of an operating coal-mining company or organization upon past, present or prospective earnings.

The application of this method gives a gross value representing that of all of the property and assets, tangible and intangible, of such company as a going concern. If from such value there be deducted the value of improvements, personal property, buildings, fixtures, structures, plant and other tangible property, the values of which may be determined by their replacement costs; and the amount of an arbitrary appraisal fixed, or agreed upon, as the value of the intangible property, goodwill, trademarks; and the values of surface real estate determined by standard real estate appraisal methods; the balance will represent the value of minable coal remaining in the property.

The proponents of such method claim that the value of a property is limited to a sum upon which the net income will be sufficient to pay a satisfactory rate of interest and to provide a sinking fund to extinguish such value at or before all of the coal is mined.

This reasoning is doubtless sound as affecting the protection of prospective investors in mortgage securities based upon such valuation, but it ignores conditions or facts which may be of equal or greater importance than the current profit or loss shown by the books.

Past Earnings Not a Fair Basis

Earnings indicate values only during the periods in which conditions are normal or approximately normal.

Earnings may fluctuate while the value of property remains unchanged.

Past earnings do not assure a continuance of like earnings in the future.

If valuation be based upon average earnings extending back over a period of years, the date of valuation is virtually set midway in the period used, so that the value is as of some year prior to the present. If the trend of earnings during such period has been downward, the value reached is too high; if upward, the value is too low. In neither case does the result represent the present value.

Therefore, valuation by past earnings is permissible only when it can be conclusively shown that no evidence proving current sales value is obtainable.

Past earnings, if large, may encourage a buyer to pay a price in excess of the value; or if small, may prevent him from purchasing even at a price much below the real value.

In addition, because the recent past includes periods of remarkable, unusual and extraordinary fluctuations in prices of materials, supplies, labor, taxes, and other items that affect earnings, both gross and net, and

because of similar unusual variations in the prices of coal and in the demand, the national method for the evaluation of coal properties is that based on sales value.

It may also be noted that during this period royalty values have been singularly free from the marked fluctuations recorded by the prices of other commodities.

Effect of Management on Earnings

In addition to the objections just given, earnings may reflect results due partly, or wholly, to superior management, or to energetic and successful salesmanship or to accidental fortuitous conditions of temporary nature—or vice versa.

Valuations based upon earnings, therefore, may often indirectly capitalize the value of "management." Although "management" may often be an asset of large value, it belongs to the intangible class, and if it is to be included in the appraisal of a property, its value could more safely be determined and stated separately from the valuation of the real property.

DISCUSSION

W. L. Affelder, Pittsburgh, Pa.—In the appraising of coal property should the value of development work be considered?

H. M. Chance.—Yes, the cost of development should be included in valuing a developed property.

R. V. Norris, Wilkes-Barre, Pa.—If, as the author says, "Sales of coal in the ground, as customarily carried out through lease under royalty agreements, are therefore the most trustworthy means available for determining the value of coal reserves," why lease coal? The coal royalties are a means of returning to the fee owner an amount larger than he would have been paid any other way. There must be a value beyond the royalty value which is the profit of the lessee; otherwise, he would not lease property.

A royalty base puts no value on a developed, or explored, property as compared with an unexplored, practically unknown property. The royalty values are not determinable with the extreme ease that would be inferred from this paper. Royalty values vary materially and are as much a matter of supply and demand as actual sales values.

I appreciate the author's position in attempting to work out a valuation based on the decisions of the courts of Pennsylvania. These courts have ruled uniformly, that the only base of value for local assessments is sales, and for taxation purposes within the state you are forced to a value based on sales, which seldom is close to the real value. In the anthracite region, values based on sales do not compare at all with the

values based on royalties and these latter vary so that they are practically worthless as a basis of value.

Royalties value do not consider the important factors of the varying character of the deposit, the markets, the cost and the profits, and all question of competition; nor the rate of development and the life of the property. Where there is great competition for coal, royalties rates may run far beyond any reason, the actual payments and value of property being kept down by low minimum tonnage requirements. In the anthracite region in the eighties, so many leases were made that the royalty rates previously paid jumped from 10 to 25, and in some cases 50, per cent. But with minimum tonnage requirements so low that the lessees were not obliged to mine actively, they were able to hold reserves so long that the actual present value was no greater than it would have been at a lower rate and much higher minimum. As a matter of fact, the value based on royalty depends more on the minimum rate of mining demanded than on the actual royalty price.

Further, a royalty value fails to take into account the effect of development on valuation. A number of individual properties undeveloped may have one value per acre; gotten together they may have a higher value; but gotten together and developed to a successful output they will have a much higher value. But the royalty would be the same in all cases. The royalty represents almost an irreducible minimum of value and not a fair market value.

An appraisal is not a mathematical calculation; it is a careful study by the engineer of the probable future. It is not based, as has been intimated, entirely on past experience. It is based, among other things, on a study of the past and a study of the future, with consideration given to reserves, changing markets, character of competition, character of deposits and probable costs and profits. It is improper to value anything that is not proved, or to figure on a management that somebody believes may be put in. But if a property is making 10 c. a ton and you know you can make 30 c. a ton on it, you may be able to buy it at a price that is high to the man who is making 10 c. and yet pay a price that is not excessive.

H. M. Chance.—The application of royalty valuation is intended for use in valuing properties leased within a reasonable time from the date in which the valuation is to be made, thus adopting the value at which the owner sold the coal.

W. L. Affelder.—When you speak of royalty as a measure of value, do you consider the royalty price or the present worth of the royalty?

H. M. Chance.—The royalty rate is used to estimate royalty annuities reduced to a present money value.

G. H. Ashley, Harrisburg, Pa.—Royalties are apt to be district, or sometimes state, wide. For example, in Indiana, 20 or 30 years ago,

royalties ran about 3 cents. In some cases, however, differencies of location, or grade of coal were recognized by a premium payment in addition to the royalties. In one case, in the Pocahontas district, that payment amounted to several hundred thousand dollars, which was more than the property was worth and was a burden. The main difficulty when treating the value of a particular property from the royalty standpoint, is that one is apt to be dealing with a royalty that extends over a whole district with little reference to individual properties.

R. V. Norris, (written discussion).—The royalty paid in any district is not a good basis for valuation and should be used only in the absence of more definite information. This is especially true of the valuation of a leasehold, for while the amount paid as royalty is a function that must be considered, it does not reflect the true value of the leasehold to the lessee, but rather the amount the lessor or owner of the property can secure for his property.

The value of a leasehold to an operating company, either for a capital investment, under the meaning of the Income Tax rulings, or for the valuation upon which securities, either bond or stock, are to be issued, is considerably more than the amount paid as royalty for said lease to the land owner. When the lessee has invested his money in the development of the lease, and with it has invested his "brains" or business experience, the leasehold has an additional value, which is determined by the amount of profit he can make over a period of years the lease will last at the rate of output of coal that the mine will be developed and equipped for.

A portion of this profit should go to the capital, in addition to the regular rate of interest together with a depletion charge of such an amount as will pay back the capital investment. This extra amount of dividend from the profit made should be for the extra risk taken in furnishing capital to a hazardous business. Another portion of this profit should go to those who furnish the "brains" and experience toward producing a successful operation, and should be distributed to these men over and above a fair salary paid for their service. A third part of the profit should be set aside as the value of the leasehold to the company and should be added as invested capital after the mine has been proved worth while, either by tests on the coal or the conditions of the lease.

If full interest is paid on the capital invested and a sinking fund set up with which to retire the said capital, a fair distribution of the profits from such a property would be one-third to the capital, one-third to the "brains" or operators, and one-third to the value of the leasehold. Ordinarily, though, one-half of the profit should be allotted to the capital, one-fourth to the operators, and one-fourth to the value of the leasehold.

The accompanying table shows how this will work out as to a value of the leasehold. The table is worked out on the basis of a profit of 10 c.

per ton on an annual production of 100,000 tons of coal; that is a profit of $10,000 per year. This$ 10,000 per year, taken for the life of the mine, and reduced to its present worth on a straight 6 per cent. basis, gives the value of the property at any fixed period as the approximate life of the mine. These figures can be increased or decreased in accordance with the tonnage produced and the profit that can reasonably be expected from the operation. And, whatever part or division of this total sum is taken as the value of leasehold, either one-third, one-fourth or one-fifth will give the value thereof.

Years	Present Worth $1.00	Yearly Profit	Amounts	Total Value	Years	Present Worth $1.00	Yearly Profit	Amounts	Total Value
1	$0.9434	$10,000	$9,434		21	$0.2942	$10,000	$2,942	
2	0.8900	10,000	8,900		22	0.2775	10,000	2,775	
3	0.8396	10,000	8,396		23	0.2618	10,000	2,618	
4	0.7921	10,000	7,921		24	0.2470	10,000	2,470	
5	0.7473	10,000	7,473	$42,124	25	0.2330	10,000	2,330	$127,837
6	0.7050	10,000	7,050		26	0.2198	10,000	2,198	
7	0.6651	10,000	6,651		27	0.2074	10,000	2,074	
8	0.6274	10,000	6,274		28	0.1956	10,000	1,956	
9	0.5919	10,000	5,919		29	0.1846	10,000	1,846	
10	0.5584	10,000	5,584	73,602	30	0.1741	10,000	1,741	137,652
11	0.5268	10,000	5,268		31	0.1643	10,000	1,643	
12	0.4970	10,000	4,976		32	0.1550	10,000	1,550	
13	0.4689	10,000	4,689		33	0.1462	10,000	1,462	
14	0.4423	10,000	4,423		34	0.1379	10,000	1,379	
15	0.4173	10,000	4,173	97,125	35	0.1301	10,000	1,301	144,967
16	0.3937	10,000	3,937		36	0.1274	10,000	1,274	
17	0.3714	10,000	3,714		37	0.1158	10,000	1,158	
18	0.3503	10,000	3,503		38	0.1092	10,000	1,092	
19	0.3305	10,000	3,305		39	0.1031	10,000	1,031	
20	0,3118	10,000	3,118	114,702	40	0.0972	10,000	1,972	150,514

For example, with a profit of 20 c. per ton, an annual output of 200,000 tons, and a life of 30 years, the profit will be twice that of the table.

The life of 30 years is $137,652 × 2 = $275,304. The output is twice that of the table; therefore, $275,304 × 2 = $550,608. If we take the value of leasehold as one-fourth of this amount, the leasehold is worth $550,608 ÷ 4 = $137,652. Any other combination can be quickly made as to the value of leasehold to fit any case.

HENRY LOUIS, Newcastle-on-Tyne, Eng. (written discussion).—The author restricts his discussion of the valuation of coal properties to properties in the hands of operating coal-mining companies, and neglects those cases which, in my experience, occur more frequently and present greater difficulties, namely the valuation of coal-mining properties that have not been attacked. The two classes of cases can no doubt be assimilated if regard is given to certain data that has not been discussed. In addition to the royalty value of the coal, it is necessary to know what

quantity of coal exists within the property. Given the quantity of coal to be mined, the prospective life of the property and the royalty value of the coal, the valuer then and only then has the data on which to base his valuation.

Even within his restricted field the author would find great help if he adopted the principle that every problem in mineral valuation is a problem in probabilities and is to be solved by means of probability calculations. No one knows beforehand the actual amount of coal in a given property, but only the most probable amount of coal which the property can be calculated to contain; this figure must form the basis of any equitable valuation.

The author distinguishes between a valuation based upon royalties and one based upon earnings, but the two methods in ultimate analysis are one and the same, because royalties are based only upon probable earnings. If coal exists under such conditions that it is absolutely impossible to earn any profit by the mining thereof, the royalty value of that coal is manifestly nil. It is unsafe to base valuation merely on the average of past earnings. If this were possible, valuation of a mineral property would be reduced to an actuarial calculation and could be performed by an accountant in his office. Although mathematical processes must be employed, mineral valuation can never become a question simply of mathematics; it must always be left to the decision of a trained mining engineer who is able to forecast, as the result of his experience, what the probable future earnings are likely to. He will take the record of past earnings into consideration, but his experience will have taught him how such earnings are likely to be modified in the future. He is not likely to capitalize the value of management because his experience will have taught him what part management has played in earnings in the past. An engineer is always entitled to assume that the future management will be the best available, and that if a property has been well managed in the past it will continue to be equally well managed in the future; however, the reverse is not true, and if a property has been badly managed in the past, the valuing engineer is entitled, or indeed bound, to fix the royalty value of the coal at what it would be under skilful management.

It goes without saying that the royalty value is affected by the quality of the coal, by the probable demand in the future for coal of this quality, by the difficulty with which the coal property can be opened up, developed and worked, by its nearness to markets, by the probable cost of labor, supplies, etc., and by a number of other circumstances. It is, therefore, perfectly right to say that the valuation of a mineral property depends, in the ultimate resort, on the probable earnings that will be obtained from it, although this probability is generally expressed in terms of royalty.

H. M. Chance (author's reply to discussion).—The intention of the author was to formulate a statement of principles applicable to the appraisal of coal properties of every kind, old and virgin, developed and undeveloped. Whether a coal property has been worked or remains unworked, whether its value is to be determined by past or prospective earnings, or by actual sales, or by offers to pay a certain price for it, the principle underlying valuation should be the same in all cases. The foregoing discussion, therefore, in part is based on a misunderstanding of the writer, for the basic principles apply to all coal, worked or unworked.

I do not agree that every problem in valuation "is a problem in probabilities," nor that valuation by royalties and valuation by earnings are "one and the same." In many cases earnings per ton are many times the maximum royalty obtainable. Leasehold operators are usually unwilling to pay as royalty more than a small part of actual, possible or probable earnings.

I must also dissent from the dictum that a valuing engineer is entitled to assume that future management will be the best available, or that if well managed in the past a property will continue to be equally well managed in the future, for this involves direct capitalization of management as a part of the property value.

Mr. Louis holds that the reverse of this is not true and that in the case of a badly-managed property, the valuing engineer is bound to fix the royalty value of the coal at what it would be under skilful management. This doctrine appears dangerous, opening the door to inflation of values with disastrous results to both engineer and clients. Prudence would seem to require that its application should be deferred until skilful management has been applied to the property and the results of such management are known.

The object of the paper was to accentuate the necessity for fixing values by known and demonstrable facts, to eliminate uncertainties, and where these exist to emphasize the necessity for confining valuation to the lowest measure of proven value.

The engineer will be venturing upon dangerous gound if he assumes a management of higher grade than that obtainable with employes of average intelligence, not including those selected because of superior intelligence. As in the operation of large units it is not possible to do much in selectively increasing efficiency, the engineer should confine his forecasts to results that can be obtained with the material at hand, endeavoring to offset the inefficiency of the average employe by improvement in methods and apparatus. Such method and apparatus must, however, be of a kind that can be operated successfully by average operatives not especially skilled in any particular kind of work. It is especially important that apparatus to be used in the mining of coal or in its preparation for market should be such as will withstand rough usage

and will perform its functions if operated by employes who do not understand its mechanical construction, so that it will continue to perform its work without mishap or delays even when operated by individuals who have never before seen such a machine or device.

Skilful management implies utilization of modern labor saving appliances, but in fixing a basis of value for the appraisal of coal properties the engineer will be justified in excluding prospective increase in earnings from the adoption of untried methods or devices, because, for the reasons above stated, these may not produce such expected increase in earnings.

From the discussion it is evident that the object of the author has not been understood. The paper was not presented as an essay on valuation methods, but was intended to clarify the possible application of methods in common use for bringing the results within the pale established by judicial decisions in the State of Pennsylvania, and in many other states and countries.

In the appraisal of property for state, county or Federal taxation, "sales value" is the governing principle in most cases; in the approval of sales made by executors and trustees the courts are largely influenced by the same method of determining whether a proposed price should be accepted or rejected; in apportioning property among heirs the same principle is recognized; and finally, buyers and sellers of property are alive to the importance of current sales value of like property.

Valuation by royalty returns was suggested as a near approach to the sales value method, where sales are not available, and as furnishing a method that may be admitted as permissible under the rules of evidence in such cases. To meet these and other tests, the paper also advocates conservatism to avoid capitalization of management, earnings, etc., and adherence to methods by which valuation is restricted to the lowest proven value.

In the absence of sales values, or of comparable leases stipulating royalty payment for the coal, valuation by earnings is discussed wholly with the object of pointing out the possibilities of error which this method may involve, and suggesting means for reducing these to a minimum.

H. N. Eavenson, Pittsburgh, Pa. (written discussion).—The writer cannot agree with the premise of the author that "sales of coal in the ground, as customarily carried out through lease, under royalty agreements, are the most trustworthy means" for determining coal property values, nor with the later statement that "royalty values have been singularly free from the marked fluctuations recorded by the price of other commodities." Taken over the whole number of coal leases in effect, the latter statement is approximately correct, largely because most of the leases were made many years ago. But with regards to leases made during the past ten years, particularly in Southern West Virginia and

in parts of Eastern Kentucky, where most of the coal is operated under lease, the statement is not true. One lease of a large area was made, about 1913, on a basis of a sliding scale of 10 per cent. of gross sales price over 90 c. per ton, and 10 c. per net ton less than that. Mines were opened and a large development made, and over the first ten year period the average royalty paid is said to have exceeded 40 c. per ton, In 1923, the property was sold and the lease was rewritten on a rate of 15 c. per net ton, with the same annual minimum per acre. In other cases, leases having a rate of 15 c. per ton were subleased at 40 c. but these royalties are not now being earned, or paid, although when the leases were made the propositions looked safe to many people. Much coal was opened during recent years (1915–1925) at royalty rates of 15 c. per ton, or thereabouts, that cannot now be worked profitably, if at all, due to thinness of seam, or poor quality. The tendency of all royalty rates in any given field is toward uniformity, regardless of quality, thickness, availability, or many other factors that influence its actual value

If a lease had no value other than that reflected in the royalty rate, it is inconceivable to the writer why anyone would make a lease and open a mine.

Mr. Norris and Professor Louis have ably stated the case of valuations based on earnings, and the writer thoroughly agrees with this idea, with the provison which must always be kept in mind, that the engineer must always use the earnings made with discretion and with full consideration of the various factors which may change them in the future, as far as they can be foreseen.

L. R. Gurley, New York, N. Y. (written discussion).—A coal seam can have no sentimental value; its only worth is as an asset of a commercial enterprise. Coal cannot be used while it is in the ground and a coal deposit that cannot be extracted and sold at a profit has no value to the owner. The fact that the property might be sold to some one ignorant of its true condition does not give it value; in the commercial sense, the only sound basis for value is profit. Therefore, "value," in connection with the appraisal of coal deposits, can be defined as "worth to an owner," either actual or prospective. The thought that value is the present worth of the future profit is not at all new. Expressed in slightly different words it is the finding of the Engineers' Advisory Committee of the United States Coal Commission, while Hoover uses the same general premise in "Principles of Mining." Finlay, too, is in agreement throughout his writings.

The great difference between "value" and "price" must not be overlooked. *Price* is the amount paid or received in an actual exchange and is the result of the various compromises, concessions and other influencing circumstances ever present in such a transaction; price might

represent value, usually it does not and can not. The reliable sales price of a coal property put on the market is an entirely different thing from its worth to the owner and is dependent on an entirely different set of factors.

The author objects to appraisals based on profits or earnings because such appraisals are made as of a past date, instead of the present, and future earnings are difficult to estimate. Besides, earnings indicate value only during normal periods and past earnings are not a guarantee of future profits. These objections might have weight if, and only if, the appraiser uses certain set formulas and extracts his factors directly from the company records without regard to the economic trend of prices and costs and without allowance for the variation in factors due to changes in operating conditions, markets or management. However, an able appraiser will carefully weigh his factors and his experience will enable him to translate the record of the past into the probabilities of the future. His result will reflect all angles of his problem and will be much nearer the true value of the property than is possible by the use of the comparative sales or other similar methods.

The equity of the "willing buyer and willing seller" theory fails completely unless the qualification is added, "with equal knowledge of the property and equal operating ability." This writer has in mind a sale between an eager willing buyer and an equally eager willing seller which, on the part of the seller, was little short of grand larceny, because his superior knowledge of the property. Such a sale can be no criterion of value. Again, let us assume that A owns a property on which he makes but 15 cents per ton on his coal. B knows that he can produce and market the same coal so as to clear 20 cents per ton, so he buys the property from "A" at 15 cents; that is clearly a willing buyer and willing seller transaction. C, however, knows that he can operate the property so as to make a profit of 25 cents a ton; he, therefore, buys the property from B at 20 cents a ton of coal remaining. That is again a willing buyer and willing seller transaction; but what is the value of the coal by that theory? It was the same coal in both transactions; the only variant is the operating ability of the three men. Certainly the "willing buyer and willing seller" definition must have further qualification to be entitled to consideration.

But having these ideal conditions of a willing buyer and a willing seller with equal knowledge of the property and equal operating ability, it is difficult to conceive of a sale. There must be duress on one side or the other or there would be no sale; their minds cannot meet in open agreement. For instance, A owns and is operating a property. He knows that under normal conditions the present value of his average future earnings is 20 cents per ton; that is how much his coal is worth to him. B, knowing these facts and being but an equal operator, will

not willingly pay twenty cents a ton for that coal, he will want to buy for less than 20 cents and make a profit or commission. That coal is not worth 20 cents to him; it is only worth 20 cents less his profit. This factor is usually called "buyer's profit" but, regardless of its name, it is ever present and establishes an insurmountable barrier to the convergence of "sales price" and "worth to an operating owner."

The paper objects to analytic appraisal because the intangible factor of management is not excluded; that objection, if sincere, is based on faulty logic. An appraisal of a coal property is based on three general factors, the amount and grade of the coal itself, the location of the deposit with respect to a market, and management. The first factor is purely physical and the other two are intangible, but all three are vital and inseparable. A variation in one of the factors causes a corresponding variation in the value; eliminate one of the factors and the entire value is eliminated. Eliminate the coal and you eliminate the value. Eliminate the market and the value is gone regardless of the quantity and quality of the coal and the management. The same rule holds in the elimination of management for coal, without management, is merely buried material and is both useless and worthless. All engineers consider the ability of the directing personnel in making an appraisal. The averaging or standardization of management gives to all similar deposits in any vicinity an equal and usually erroneous unit value.

Some operators achieve their results with less expense than others and therefore make greater profits. Their properties as revenue producers are worth more than the same or similar properties under less efficient management. Eliminate the variability of that factor from the appraisal and the result is not a true "owners' worth."

Relation of Ash Composition to the Uses of Coal*

By A. C. Fieldner† and W. A. Selvig,‡ Pittsburgh, Pa.

(New York Meeting, February, 1926)

Ash in coal has always been regarded as an undesirable substance, as the heat content of a coal decreases in direct proportion to its ash content. It represents so much inert material that has to be transported, handled when the coal is burned, and finally disposed of. W. R. Roberts, in an address before the Engineers' Society of Western Pennsylvania, Pittsburgh, on Sept. 29, 1925, said that on an annual output of five hundred million tons of coal, a reduction of 3 per cent. of ash means fifteen million tons of refuse and a decrease of $30,000,000 in freight charges. Ash also often prevents efficient utilization of the coal through the formation of clinker caused by the melting of the ash constituents when subjected to heat. The mineral constituents of coal may be of such nature as to make the coal unsuited for specific uses, as for instance, high-sulfur coal for making metallurgical coke.

While the usual proximate analysis of coal gives the amount of ash, it gives no information as to the nature and composition of the ash nor of the mineral constituents from which the ash is produced. For the efficient washing of coal to remove ash-forming constituents it is desirable to know the composition and distribution of the ash-forming constituents. Obviously such information is also desirable for the most efficient utilization of coal.

Nature and Composition of Coal Ash

Coal ash may be defined as the inorganic residue remaining after complete ignition of coal. It is derived from the mineral constituents of the coal. The ash-forming constituents may be roughly classified as inherent impurities and extraneous impurities. The former are mixed intimately with the coal substance, and are derived either from the original coal-forming material or from external sources such as sedimentation and precipitation while the vegetal coal-forming plant remains accumulated. The latter are present in the coal bed as partings, bands or nodules of pyrite, slate, shale, calcite, bone, etc. These are formed either during the laying down of the coal bed, or subsequently. Frag-

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ments of roof and floor that become mechanically mixed with the coal in the process of mining also contribute to the extraneous impurities.

The percentage of ash derived from the intimately mixed impurities may be determined by selecting a sample of clean lumps of homogeneous coal, or better, by placing the coarsely crushed coal in a solution of zinc chloride with a specific gravity of 1.35. The coal will float, and the heavier extraneous impurities will sink. For some samples the percentage of inherent ash is so low that the ash must have come exclusively from the original coal-forming vegetation. A notable example of coal exceptionally low in ash is found in the Sewell bed of West Virginia; analysis of several mine samples taken from this bed near Beckley, in Raleigh county, gave between 1.5 and 2 per cent. ash. In some samples the intimately mixed impurities may reach a high proportion, as in bony coals, which yield 25 to 40 per cent. ash. Certain coals have thin flakes of calcite, gypsum, silica, or clay in their minute joints or cleavage planes and some have veinlets of pyrite. English investigators[1] call the white partings "ankerites," and usually find them to consist of carbonates of lime, magnesia and iron. These impurities are not intimately mixed with the coal substance, yet they cannot be separated from the coal by ordinary methods of preparation. Microscopic crystals of pyrite are also found distributed in some coals to such an extent that washing produces no appreciable lowering of the sulfur content.

The percentage of ash derived from extraneous impurities varies considerably, depending upon the number and size of the partings in the bed, the possibility of separating these from the coal, and the care with which the coal is mined. The possibility of improving the quality of coal by washing may be determined by the float-and-sink test already described.

As a rule, coal ash as determined weighs less than the mineral matter from which it is produced. This is due to the loss of volatile constituents during ignition. Shale and clay lose their water of hydration, the carbonates are more or less decomposed and give off carbon dioxide; and the pyrite or marcasite is changed to ferric oxide and gives off sulfur dioxide, either to the atmosphere or to the free calcium oxide that has been formed from the carbonate. In coals that contain appreciable amounts of calcium carbonate, a large proportion of the sulfur may be retained in the ash as calcium sulphate. Parr[2] has proposed a method of computing the approximate weight of the inorganic matter in coals that contain calcium carbonate and iron pyrite by adding corrections to the weight of ash

[1] F. S. Sinnatt, A. Grounds and F. Bayley: The Inorganic Constituents of Coal with Special Reference to Lancashire Seams. *Jnl.* Soc. Chem. Ind. (London, 1921) **40**, 1T.

[2] S. W. Parr: Preliminary Report of Committee on Coal Analysis. *Ind. & Eng. Chem.* (1913) **5**, 523.

obtained by ignition. Such methods of computation are necessarily approximations only, as it is assumed that all of the sulfur is present as pyrite, and an arbitrary factor for the water of hydration of clayey matter is used. The principal application of "corrected ash" values is in computing the actual coal substance or combustible matter of coal for comparing ultimate analyses and heating values on this basis. For technical purposes the uncorrected ash is reported as determined.

Coal ash is composed largely of compounds of silica, alumina, lime, and iron, with smaller quantities of magnesia, titanium, and alkali compounds. The chemical composition varies widely but in general comes within the following limits:

TYPICAL LIMITS OF COAL-ASH ANALYSES

	Per Cent.		Per Cent.
Silica, SiO_2	40–60	Calcium oxide, CaO	1–15
Alumina, Al_2O_3	20–35	Magnesium oxide, MgO	0.5–4
Ferric oxide, Fe_2O_3	5–25	Titanium oxide, TiO_2	0.5–3
		Alkalies, $Na_2O + K_2O$	1–4

Table 1 has been compiled by Marson and Cobb[3] from the work of various investigators, and shows the probable forms in which ash constituents exist in coal:

TABLE 1.—*Inorganic Constituents of Coal*

Inorganic Constituents	Forms in Coal	
Silicon	Silicates, sand	
Aluminum	Alumina in combination with silica	
Iron	Pyrite and marcasite (sulfide)	
	Ferrous oxide Ferrous carbonate Ferrous sulfate Ferric oxide Ferric sulfate "Organic" iron Iron silicates	in small quantities
Calcium	Lime, carbonate, sulfate, silicates	
Magnesium Sodium and potassium Manganese	Carbonate, silicates Silicates, carbonates, chlorides Carbonate, silicates	in small quantities
Sulfur (inorganic)	Pyrite and marcasite	
	Ferrous sulfate Ferric sulfate Calcium sulfate	in small quantities
Phosphorus	Phosphates	

[3] C. B. Marson and J. W. Cobb: Influence of the Ash Constituents in the Carbonization and Gasification of Coal, with Special Reference to Nitrogen and Sulfur. Part I—Preparation and Preliminary Examination of Special Cokes. *Gas Jnl.* (London, 1925) **171**, 39.

Sulfur Forms in Coal

Powell[4] made a study of the analysis of sulfur forms in coal, and found that considerable sulfur may occur in other forms than that combined with iron as pyrite or marcasite. Table 2 shows the forms in which sulfur existed in six samples of coal from various fields of the United States.

Table 2.—*Form of Sulfur in Six Different Samples of Coal*

Source of Coal Bed	Sulfur Form, Per Cent.			
	Pyritic	Sulfate	Organic	Total
Upper Freeport, Pa.	0.47	0.07	0.62	1.16
Pittsburgh, Pa.	0.79	0.23	0.66	1.68
Pocahontas, W. Va.	0.08	0.01	0.46	0.55
Elkhorn, Ky.	0.13	0.04	0.51	0.68
Coal Creek, Tenn.	1.75	0.71	1.78	4.24
Cherokee, Kans.	1.99	0.32	0.71	3.02

It will be noted that considerable sulfate sulfur is present in some of the coals listed. Doubtless this is due to the fact that the samples had stood for some time. As a rule, freshly mined coal contains only very small amounts of sufate sulfur.

Yancey and Fraser[5] studied the distribution of pyritic and organic sulfur in coal as it occurs in various sections, layers, or benches of a coal bed. Variation of total sulfur between sections or benches of the same bed at a given place, in any except low-sulfur coals, may be marked. This is due principally to the heterogeneous or "spotted" distribution of iron pyrite. More or less of the pyrite, depending upon its physical form, can be removed by coal-washing methods. It was found that many coals contained a relatively high proportion of sulfur in the organic form. In 13 of 34 bench samples from one bed the organic sulfur exceeded the pyritic sulfur. In comparison with the large variations of pyritic sulfur in the vertical space of the bed, the distribution of organic sulfur was uniform. Washing tests on seven samples of run-of-mine coal from one mine gave an appreciable reduction of pyritic sulfur, but did not remove the organic sulfur due to its uniform distribution. If the organic sulfur had been segregated with or concentrated around pieces of pyrite, bone coal, or shale it would have been possible to remove it with these impurities in the washing operation. Some coals have an organic sulfur content sufficiently high to limit seriously the extent to which these coals can be cleaned of sulfur by washing.

[4] A. R. Powell: The Analysis of Sulfur Forms in Coal. U. S. Bur. of Mines, *Tech. Paper* 254 (1921), 21 pp.

[5] H. F. Yancey, and Thomas Fraser: The Distribution of the Forms of Sulfur in the Coal Bed. *Ind. & Eng. Chem.*, (1921) **13**, 33.

Sulfur Forms in Coke

The reactions of the sulfur compounds of coal are of especial interest in the production of metallurgical coke, particularly from the viewpoint of possible methods of desulfurization of coke. Powell[6] found that there are usually four different forms of sulfur in coke as it is received by the consumer—namely, as ferrous sulfide, sulfate sulfur, free adsorbed sulfur, and solid-solution sulfur. A high percentage will be found as ferrous sulfide and solid-solution sulfur. Small quantities of sulfates and free adsorbed sulfur will be found, but they may be absent entirely, especially if the coke has been made by prolonged heating and has been quenched quickly. By carbonization tests on a variety of coals Powell concluded that these sulfur reactions[7] may occur:

Complete decomposition of the pyrite to form pyrrhotite and hydrogen sulfide. This reaction begins at 300° C., is complete at 600° C., and reaches its maximum between 400 and 500° C.

Reduction of sulfates to sulfides. This reaction is complete at 600° C.

Decomposition of one-quarter to one-third of the organic sulfur to form hydrogen sulfide. This occurs for the most part below 500° C.

Decomposition of a small part of the organic sulfur to form volatile organic sulfur compounds, most of which find their way into the tar. This decomposition occurs at the lower temperature of the coking process.

Disappearance of a portion of the pyrrhotite, the sulfur apparently entering into combination with the carbon. This reaction seems to be most active at 500° C. or higher.

Powell concludes that his investigations indicate that the total sulfur of the coal is the most important factor affecting the sulfur content of the coke, and that the relative amounts of sulfur forms present do not affect it materially. He also concludes that when coke cools, even with limited access to the air, oxidation of ferrous sulfide occurs, according to the reaction:

$$4FeS + 3O_2 = 2Fe_2O_3 + 4S.$$

When coke is quenched quickly, as in ordinary coke manufacture, this decomposition of ferrous sulfide is not complete, because of the speed with which the temperature of the coke is carried below that for effective oxidation.

Iron Forms in Coal

Sinnatt and Simpkin[8] investigated the analysis of iron forms in coal and found that the iron may occur in at least five distinct forms—namely,

[6] A. R. Powell: The Forms of Sulfur in Coke. *Jnl.* Amer. Chem. Soc. (1923), **45**, 1–15.

[7] A. R. Powell: Some Factors Effecting the Sulfur Content of Coke and Gas in the Carbonization of Coal. *Jnl.* Ind. & Eng. Chem. (1921), **13**, 33.

[8] F. S. Sinnatt and N. Simpkin: The Iron in Coal. *Jnl.* Soc. Chem. Ind. (London, 1922) **41**, 164T.

in the ankerite (calcium, magnesium, and iron carbonate), in iron oxide or carbonate, water-soluble iron salts, silicate, and as pyrite. Analysis of a number of English coals for iron forms showed that the manner in which the iron occurs in different coals varies over wide limits. The iron present as pyrite represented the predominating variety, and in some coals practically all the iron may be present in this form. Silicate iron was found to be completely absent in some coals, while in others it comprised as much as 10 per cent. of the total iron.

Gas and Coking Coal

It is apparent that the amount of ash present in coal must be considered together with composition of the ash in relation to the influence of the ash to the uses of coal. The standard specifications for gas and coking coals of the American Society for Testing Materials[9] state that the ash of either gas or coking coal should not exceed 9 per cent. The sulfur content of coke from gas coal is limited to 1.5 per cent. the sulfur content of metallurgical coke is limited to 1.0 per cent. in the case of foundry coke and 1.3 per cent. in the case of blast-furnace coke. The limits of sulfur in domestic gas is placed at 30 grains of sulfur, in the form of compounds other than hydrogen sulfide, per 1000 cu. ft. of gas. These limitations as to sulfur necessarily exclude the use of high-sulfur coals for making gas or coke.

High-ash in blast-furnace coke is very undesirable, as additional limestone is required for slagging the silica and alumina of the coke ash, and necessitates not only an increase in the amount of limestone used but also an increase of coke to produce the additional heat required. Read, Joseph, and Royster[10] estimate, from data obtained on 30 blast furnaces, that each additional pound of slag requires 0.45 pound of fixed carbon, which is equivalent to 0.52 of a pound of coke (86 per cent. fixed carbon). This estimate derived from the effect of silica in iron ore should also be applicable to other sources of silica, as that of the coke ash, and also to the alumina of the coke ash. Sweetser[11] states that coke made from coal containing 1 per cent. extra ash is worth 30 cents a ton less for production of pig iron.

Some investigators have considered that the hardness of coke is due to reactions during the coking process of a portion of the ash constituents of the coal with the carbon to form carbon silicides, also that iron silicide is formed and contributes to the hardness. However, this supposition is not generally accepted.

[9] Am. Soc. Testing Materials, *Standards*, (1924), 973.

[10] T. T. Read, T. L. Joseph and P. H. Royster: The Effect of Silica in Iron Ore on Cost of Pig Iron Production. U. S. Bur. of Mines, *Rep. of Investigations*, Ser. No. 2560, 1924.

[11] R. H. Sweetser: Wastefulness of High Ash in Coal. *Blast Furnace & Steel Plant*, (1924) **12**, 103.

Lessing and Banks,[12] and Lessing[13] studied the catalytic effect of added mineral constituents to coal in the production of coke on a laboratory scale. They found that a larger percentage of coke was produced from the same coal on the addition of small quantities of mineral compounds, and that the catalysts which increased the coke yield also tended to raise the percentage of carbon in the coke obtained. They conclude that the influence of ash constituents or added inorganic compounds is of fundamental character, in that they are capable of directing the course of the primary decomposition of the coal substance during the coking process.

Marson and Cobb[14] incorporated various inorganic oxides with a coal having less than 1 per cent. of ash, and coked the resultant mixtures in the laboratory at temperatures of 500 and 800° C. Mixtures of coal containing 5 per cent. of silica, aluminum oxide, iron oxide (Fe_2O_3), calcium oxide as CaO and also as $CaCO_3$, and sodium oxide as Na_2CO_3 and also as NaOH, were used in these tests. The cokes prepared at 500° C. from the original coal and the mixtures of coal with silica, aluminum oxide, and calcium oxide, were all very similar in appearance—they were swollen, spongy, and not homogeneous. The cokes prepared at 800° C. from the original coal and the oxides mentioned were closer grained in structure than the corresponding 500° C. cokes, but were still spongy, with many large pores, and were not homogeneous. The addition of sodium hydrate destroyed the coking property of the coal. The cokes prepared from the mixtures of coal with iron oxide (Fe_2O_3) and sodium oxide added as Na_2CO_3 were markedly different in physical character from the coke produced from the untreated coal. The cokes produced from the mixture of coal with iron oxide were compact and homogeneous. Sodium oxide added as Na_2CO_3 also gave cokes that differed materially from that of the untreated coal as they were fine-grained, homogeneous and free from large pores. Crushing-strength tests showed that the strength of the coke had been raised by the addition of iron oxide, and very much more raised by the addition of sodium carbonate. Analysis of the cokes showed that there were no differences in nitrogen content of the cokes prepared at 500° C., but at 800° C. it was found that the additions of Fe_2O_3, CaO, and $CaCO_3$ had decreased the amount of N_2 retained by the coke, while sodium hydrate and carbonate had increased it. The other oxide additions were without effect. Iron oxide and lime retained the sulfur held by the 500° C. coke, and to a greater extent in the

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[12] R. Lessing and M. A. L. Banks: The Influence of Catalysts on Carbonization. *Jnl.* Chem. Soc., (London, 1924) **125,** 2344.

[13] R. Lessing: Influence of Ash Constituents on the Coking Process. *Gas Jnl.* (London), (1925) **71,** 541.

[14] C. B. Marson and J. W. Cobb: Op. cit.

800° C. coke; NaOH and Na_2CO_3 tended to retain the sulfur in the 800° C. cokes; but this effect was not evident in the 500° C. cokes.

Bähr[15] determined the ignition temperature in air, the reaction temperature with CO_2, and combustibility in air of laboratory samples of coke prepared from ash-free coal and also from the same coal mixed with 3 per cent. and 6 per cent. ferric oxide (Fe_2O_3). The ignition point was taken as the lowest temperature at which air reacted with the coke to form CO_2; the reaction temperature with CO_2 was defined as the lowest temperature at which CO began to form on passing CO_2 through the coke; the combustibility was determined by passing air through the coke heated to 950° C. and then analyzing the products of combustion for CO and CO_2. The formula used to calculate combustibility was as follows:

$$\text{Combustibility} = \frac{100 \text{ (per cent. CO)}}{\text{per cent. CO} + 2 \text{ (per cent. } CO_2)}.$$

Bähr found that the ignition points and reaction temperatures of the cokes made from mixtures of coal with iron oxide did not differ materially from that made from the pure coal; however, he found considerable difference in combustibility as calculated by the above formula. The figure obtained for combustibility of the coke from the pure coal was 17.5 per cent., that of the coke from the same coal mixed with 3 per cent. iron oxide was 58.4 per cent., and that of the coke from the coal mixed with 6 per cent. iron oxide was 70.5 per cent. This shows that iron oxide has considerable influence in increasing the combustibility of coke in air.

Coal for Steam Purposes

The softening or fusing temperature of coal ash is of particular interest because of clinker and slag trouble often experienced while burning coal. The tendency in recent years to large boiler and stoker units, operated at high ratings, has increased the clinkering trouble and has caused considerable concern to power-plant engineers. Clinker and slag formation is the result of melting of the ash constituents of the coal burned when subjected to the high temperatures attained in the furnaces.

The U. S. Bureau of Mines has recognized the desirability of investigating the clinker problem, and as a first step in a study of clinker formation has investigated laboratory methods of determining the fusibility of coal ash and the effect of various factors on the softening temperature.[16] A standard method was developed which was subsequently accepted by the American Society for Testing Material as one of its standard tests,[17] and using this method a general survey was made of the fusibility of ash

[15] Von H. Bähr: Die Untersuchung von Kohlen und Koks in Hinblick auf die Herstellung einer bewussten Kokseigenschaft. *Brennstoff-Chemie*, (1924) Bd. **5**, 384.

[16] A. C. Fieldner, A. E. Hall and A. L. Feild: The Fusibility of Coal Ash and the Determination of the Softening Temperature. U. S. Bur. of Mines, *Bull.* 129 (1918), 146 pp.

[17] Am. Soc. Testing Materials *Standards* (1924), 994.

from various coals of the United States.[18] In general, the softening temperature of coal ash from the coals of the United States ranges from 1900 to 3100° F. The order of fusibility of ash may be expressed by subdividing this range of softening temperature into three groups, as follows:

Class 1—refractory ash, softening above 2600° F.

Class 2—ash of medium fusibility, softening between 2200 and 2600° F.

Class 3—easily fusible ash, softening below 2200° F.

Class 1 may be considered as practically non-clinkering. Class 3 will form considerable clinker, which may, in many cases of high furnace temperatures, spread over the grates. The clinkering characteristics of class 2 will depend upon the furnace temperatures, the kind of stoker, and the distribution of the ash-forming constituents in the coal. The experience of a number of investigators[19] has shown that the tendency toward clinker formation is roughly proportional to the softening temperature as determined in the laboratory.

The coordination of laboratory tests to clinkering tendencies of coal is a difficult problem and has to be worked out. A laboratory test on a small sample of intimately mixed ash differs considerably from conditions that generally exist when the coal is burned; as a rule, there is no such intimate mixing of the ash-forming constituents when the coal is actually fired. Some coals have been reported of apparently the same ash-fusing temperature which have markedly different clinkering properties. The Bureau of Mines is starting an experimental study of clinker formation in an experimental furnace in which conditions can be accurately controlled. It is proposed to run a series of coals of different ash-fusing temperatures ranging from the lowest to the highest in the scale, and correlate the clinkering properties with the results of various laboratory fusion tests. The nature and distribution of the ash-forming constituents of the coals will be carefully investigated.

The fusibility of coal ash is affected by the ratio of the silica to the bases present, the particular bases, and the percentage of alumina present. Ash high in silica is not readily fusible, and as a rule, ash that is low in iron is highly siliceous and not readily fusible. Ash from coals high in pyrite, FeS_2, is necessarily high in iron, and the ratio between the bases and

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[18] W. A. Selvig and A. C. Fieldner: Fusibility of Ash from Coals of the United States. U. S. Bureau of Mines, *Bull.* (1922), 209.

[19] E. G. Bailey and W. B. Calkins: Fusing Temperatures of Coal Ash. *Power,* (1910) **32,** 1978.

E. G. Bailey: The Fusing Temperature of Coal Ash. *Power,* (1911) **34,** 802.

E. B. Ricketts: Report of Committee D-5 on Coal and Coke. *Proc.* Am. Soc. Testing Materials (1923), **23-I,** 413.

J. F. Barkley: Ash-softening Temperatures and Clinkering of Coals in a Boiler Furnace. U. S. Bur. of Mines *Repts. of Investigations* Ser. No. 2630 (1924), 3.

silica is often such that easily fusible compounds may be formed. As a rule, coals containing considerable sulfur in the form of pyrite are likely to give clinker trouble. Under conditions existing in the fuel bed of a furnace the iron of the pyrite is likely to be converted to ferrous silicates, which fuse at comparatively low temperatures. Sulfur, in combination with iron as pyrite, is a common constituent of coal, and is sometimes held responsible for clinkering. It is not, as a matter of fact, the element sulfur which produces the trouble but rather the element iron. The sulfur which exists in forms other than iron pyrite, such as in organic combination with the coal substance, and in the form of sulfate, has little or no influence on clinkering. Some coals extremely low in sufur give readily fusible ash because the other mineral constituents present are of such nature and proportion that readily fusible compounds may be formed.

The calculation of degree of fusibility from the ash analysis gives only a rough approximation on account of the large number of constituents present in the ash and the varying proportions of these constituents.

If the ash-forming minerals consist principally of siliceous and argillaceous material the ash is not readily fusible, but if they contain considerable amounts of calcite, gypsum, or pyrite, the ash is very likely to be readily fusible. Such extraneous impurities as slate, sandstone, or shale tend to raise the fusibility of the ash. It is very doubtful whether large lumps of extraneous impurities have much, if any, influence on slagging in the furnace. Large pieces of slate are usually found unfused in the clinker and refuse. Table 3 gives analyses of ash from five American coals ranging from a very fusible ash to a highly refractory ash.

Undoubtedly the composition of coal ash has considerable bearing on the slagging of firebrick in steam boiler furnaces. The failure of the

TABLE 3.—*Chemical Analyses of Ash from Five Coals Covering a Wide Range of Fusibility*

Sample No.[a]	Softening Temperature, °C.	Analyses of Ash, Per Cent.								
		SiO_2	Al_2O_3[b]	Fe_2O_3	TiO_2	CaO	MgO	Na_2O	K_2O	SO_3
1	1130	30.7	19.6	18.9	1.1	11.3	3.7	1.9	0.5	12.2
2	1270	46.2	22.9	7.7	1.0	10.1	1.6	0.7	0.8	8.9
3	1370	49.7	26.8	11.4	1.2	4.2	0.8	1.6	1.3	2.5
4	1500	51.0	30.9	10.7	1.9	2.1	0.9	1.0	0.4	0.6
5	1590	58.5	30.6	4.2	1.8	2.0	0.4	0.7	0.9	0.9

[a] Sample No. 1, sub-bituminous coal, No. 3 bed, Montana; No. 2, bituminous coal, No. 6 bed, Illinois; No. 3, bituminous coal, Pittsburgh bed, Pennsylvania; No. 4, semi-bituminous coal, Pocahontas No. 3 bed, West Virginia; No. 5, bituminous coal, Dean bed, Kentucky.

[b] P_2O_5 included with Al_2O_3.

refractory brick is of course not only due to slagging with the coal ash but also to erosion. The slagging effect may be considered to be due to the furnace temperatures, the nature of the coal ash, and the composition of the refractory with which the coal ash comes in contact. Coal ash high in basic constituents may cause considerable slagging of furnace linings of an acid character. Brennan[20] made some interesting fusion tests with various firebricks when mixed with coal ashes of varying composition. The fusion tests were made on cones consisting of 70 per cent. of ground firebrick and 30 per cent. coal ash. He found considerable difference in fusibility when the firebricks were mixed with the different coal ashes. The actual performance of a number of the bricks tested was observed for about two years, and in a majority of cases the actual life of the furnace linings followed closely the experimental results obtained in the laboratory with the firebrick and coal-ash mixtures.

It is seldom that objections are raised to a coal on account of its low ash content, however, Flagg[21] has described stoker trouble experienced on burning a high-grade West Virginia coal containing only 4.3 per cent. ash. The tops of the grate bars reached a dark red to cherry red heat and serious damage resulted thereto. He attributed this to insufficient ash to protect the grate bars properly, and the trouble was at once remedied by substituting coal of higher ash content. Flagg also states that the composition of coal ash must have a marked influence on its heat conductivity, and cites certain coals of southeastern Kansas which are high in ash, the ashes of which are high in iron content, which coals in many cases damage the grate when burned on chain grates.

Powdered Coal

The use of powdered coal for steam purposes involves consideration of the fusibility of the ash, which has been discussed above. The trouble experienced is due to slag formation on the cool surfaces of the water tubes and the slagging of the refractories of the furnace chamber.

In the case of powdered coal for cement kilns the ash of the coal becomes a part of the clinker, but this is not considered objectionable as the impurities of the coal are approximately of the same composition as the cement.

Powdered coal for iron and steel metallurgy should be low in sulfur. According to Gadd,[22] powdered coal used in heating and puddling furnaces should not have more than 9.5 per cent. ash and the sulfur con-

[20] J. J. Brennan: The Effect of Coal Ash on Refractories. *Combustion* (1924) **10**, 418–422.

[21] S. W. Flagg: Low Ash Not Always Desirable. *Power*, (1922) **55**, 328.

[22] C. J. Gadd: Powdered Coal in Metallurgy. *Jnl.* Franklin Institute (1916), **182**, 329.

tent of the coal should not exceed 1 per cent. For open-hearth furnaces the ash is preferably limited to 6 per cent. and the sulfur to 1 per cent. Low ash is considered desirable as there is likely to be slagging in the combustion chamber, in the hearth, and in the flues.

Manufacture of Water Gas

Coke and anthracite have been the standard fuels for the manufacture of water gas. During the past few years many attempts have been made to use bituminous coal, with varying success. The principal use of water gas in cities and towns is for lighting and for domestic heating purposes, and for these purposes it is usually mixed with illuminating gas. It is probable that bituminous coal will be used extensively in the future for the manufacture of water gas, with proper design of the generator sets. Odell[23] made a study of the possibility of substituting Indiana and Illinois coal for coke in water-gas generator sets and suggests a number of operating possibilities pertaining to design of generator sets that might improve the results obtained from using bituminous coking coal as a fuel. The use of high-ash coal, the ash of which is readily fusible, offers considerable difficulties because of clinker formation in the generator which tends to obstruct the passage of air and steam through the fuel bed and thereby decreases the capacity of the generator. Odell suggests methods of operating possibilities that might improve the results obtained with the use of bituminous coking coal. He suggests the use of an efficient fuel spreader to prevent the accumulation of fine coal in the center of the generator, which increases the air resistance and also decreases it near the walls, thus causing higher temperatures at the walls with the formation of clinker on the wall.

Smithing Coal

Coal with readily fusible ash is not desirable for smithing use as the resultant clinker makes it difficult to keep the fire clean, and if the ash content is high the clinker formed will tend to obstruct the forge blast. High-sulfur coals are considered objectionable because of the formation of "red-shortness" in the iron. Sloman[24] made a number of forge tests for "red-shortness" with various coals containing as high as 1.36 per cent. sulfur, and did not find any indications of "red-shortness" in the iron. He concluded that a good smithing coal should not have more than 7 per cent. ash and that low sulfur is desirable. However, a coal containing even more than 1 per cent. sulfur may be successfully used.

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[23] W. W. Odell: Water-gas Apparatus and the Use of Central District Coal as Generator Fuel. U. S. Bureau of Mines, *Tech. Paper* 246 (1921), 28 pp.

[24] H. J. Sloman: An Investigation of the Properties of Smithing Coals. Pennsylvania State College, Mining Experiment Station, *Bull.* 1 (1922) 16 pp.

Domestic Use

All coals are used more or less for domestic purposes. Low ash is, of course, desirable both from the standpoint of heating value of the coal and the disagreeable feature of handling and removal of ashes. The composition of the ash is of no special importance for domestic heating, except in connection with clinker trouble with coals of low-ash fusibility. Coal used for making domestic coke should have an ash fusibility of not less than 2200° F., otherwise considerable clinker trouble may be experienced on burning the coke. Table 4 is given by Breckenridge and Flagg[25] as showing the proportion of ash and the average and maximum proportions of refuse for some typical coals:

Table 4.—*Non-Combustible Matter in Anthracite and Bituminous Coal*

Coal	Average Proportion of Ash in Dry Coal, Per Cent.	Refuse	
		Average, Per Cent.	Maximum Allowable, Per Cent.
Pennsylvania anthracite:			
Egg	10	13	15
Stove	12	15	17
Chestnut	14	16	18
Pea	16	18	20
No. 1 and No. 2 buckwheat	18	21	23
Yard pea	15	17	19
Bituminous:			
Eastern	8–14	10–17	12–19
Indiana and Illinois	8–16	10–19	12–21
Southern Kansas	10–13	12–16	14–18

Summary

In recent years, the inorganic constituents of coal have received considerable attention by American and foreign investigators. A knowledge of the exact nature and distribution of the ash-forming constituents of coal and their reactions when coal is burned or coked is important for the most efficient utilization of coal. Such information has a direct bearing on clinker and slag trouble often experienced in burning coal, on removing impurities by coal-washing methods, on coke and gas manufacture, and the use of coal and coke for metallurgical purposes.

DISCUSSION

A. St. John, New York, N. Y.—Many bituminous coals, when fractured, show a yellowish mark; when a fragment was examined, the X-rays showed a marked distribution of ash in certain layers. The

[25] L. P. Breckenridge, and S. B. Flagg: Saving Fuel in Heating a House. U. S. Bur. of Mines *Tech. Paper* 97 (1917), 24.

fragments were arranged so that these layers were horizontal, then a careful examination showed they were the horizontal cleavages. A preliminary investigation of the distribution of the ash itself—a very rough investigation, of course—would give some information that might be of considerable value.

A feature in connection with the X-ray examination of coal developed by MacClaren and others in England relates to the evaluation of the intrinsic and removable ash. I am not persuaded that the results are any better or any more readily secured than those obtained by customary test methods, but the method is worth considering.

T. M. Chance, Philadelphia, Pa.—The washing of high sulfur coals has at times increased the sulfur content of the washed coal over that of the raw coal; this is due to the organic sulfur present. Thus, if there is 10 per cent. of removable refuse in the coal and the total raw coal contains 4 per cent. organic sulfur, this 4 per cent. will remain in the 90 per cent. of washed coal recovered so that the organic sulfur content of the washed coal will be in excess of 4.5 per cent. In one case, the raw coal contained 2.2 per cent. total sulfur (organic and pyritic). The washed coal appeared to be perfectly clean and contained but traces of visible pyrite. We, however, concluded that it was heavily charged with finely divided pyrite because the washed coal, instead of running 1.7 to 1.8 per cent. sulfur, contained 2.8 per cent. sulfur. On determining the total sulfur that was combined with iron and as gypsum we found that most of the sulfur, content was organic and thus the increase in sulfur after cleaning.

H. M. Chance, Philadelphia, Pa.—The term "organic" sulfur has long been applied to sulfur that is not combined with iron, lime or other inorganic bases. In 1879, Drown[26] suggested the term "organic" and presented an analytical method for determining sulfur not combined with inorganic bases. Kimball[27] expressed grave doubt as to the presence of sulfur not combined with inorganic bases and commented on Report M.M. of the Geological Survey of Pennsylvania which had just been printed and which contained analyses of 25 coals containing such excess sulfur as not sufficient to establish the presence of such excess.

Report M.M. was the work of Andrew S. McCreath, and is but one of his many achievements in showing the way to better and more accurate knowledge of the chemical composition of coals, ores, limestones and other materials with which the metallurgist has to deal. He discovered and proved the presence of what, for want of a better term, he called "free sulfur." He was slow to commit himself definitely to the theory that this sulfur was held in some form of chemical combination with the organic

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[26] Thomas M. Drown: Determination of Sulphur in Sulphides and in Coal and Coke. *Trans.* (1879–80) **8**, 569.

[27] James P. Kimball: Relations of Sulphur in Coal and Coke. *Trans.* **8**, 181.

carbon or hydrocarbons of the coal substance, doubtless because he thought that he did not have absolute or definite proof that such chemical combination actually existed and, as a matter of scientific accuracy, he preferred to use the term "free sulfur" as a means of positively indicating that this sulfur was not combined with any inorganic base.

The relation between pyritic sulfur and organic sulfur is often subject to such abrupt changes that it becomes necessary to determine by thorough sampling whether the sulfur is largely in the form of pyrite or not, because unless most of the sulfur exists as pyrite it is useless to expect to remove it by any washing process. It has always been a matter of interest to know whether sulfur present in such "organic form" produces the same objectionable difficulties in the operation of blast furnaces as pyritic sulfur or not.

J. R. Campbell, Scottdale, Pa.—Is not all sulfur that enters the blas furnace ultimately assimilated by the iron, so that organic sulfur is just as objectionable as any other form?

A. C. Fieldner.—In answering this question consideration must be given to the fact that sulfur forms in coal are changed in the coking process. The pyritic sulfur of the coal decomposes according to the reaction, $FeS_2 = FeS + S$. Some of the resulting ferrous sulfide may react with water vapor to form hydrogen sulfide; also most of the free sulfur is changed to hydrogen sulfide by the action of the hydrogen in the gas, especially when the gas contains more than 50 per cent. hydrogen. Most of the hydrogen sulfide passes out of the coke oven with the gas, but some hydrogen sulfide, as well as free sulfur, is adsorbed in the coke, and still another part is in solid solution.

Organic sulfur compounds in coal are decomposed in the coking process and the sulfur is largely retained in the coke as adsorbed free sulfur and solid solution sulfur. Following are some typical analyses of the sulfur forms in coke by Powell and Thompson[28] of the Bureau of Mines:

Results of Sampling Tests

Sulfur as	Ohio Experimental Coke, Per Cent.	Illinois Metallurgical Coke, Per Cent.	Pennsylvania Metallurgical Coke, Per Cent.	Pennsylvania Gas House Coke, Per Cent.
Ferrous sulfide	0.39	0.22	0.12	0.15
Sulfates	0.13	0.00	0.00	0.00
Adsorbed free sulfur	0.98	0.04	0.03	0.58
Solid solution sulfur	1.78	0.70	0.55	0.18
Total sulfur	3.28	0.96	0.70	0.91

[28] A. R. Powell and John H. Thompson: A Study of the Desulphurization of Coke by Steam. *Bull.* No. 7, Carnegie Institute of Technology, (1923) 34.

It will be noted that most of the sulfur in these cokes was adsorbed or dissolved in the coke substance; the remainder was ferrous sulfides.

Powell[29] has shown how to determine the forms of sulfur in coal and coke, and the reactions of sulfur in the coking process. Research is now needed on the behavior of the various forms of coke sulfur in the blast-furnace before one can say whether ferrous sulfide sulfur in coke is more, or less objectionable than the adsorbed or solid solution sulfur. Powell outlined such a study in a paper[30] in 1923; unfortunately industrial competition for Dr. Powell's services took him away from the Bureau of Mines shortly thereafter and the proposed study was never taken up.

Organic sulfur in coal is more objectionable than pyritic sulfur as most of it is retained in the coke, while over half of the pyritic sulfur is removed in the gas.

J. R. Campbell, Scottdale, Pa.—Sulfur is sulfur to the blast furnace man. He is not much interested in whether it is a form of sulfide or organic sulfur.

H. M. Chance.—Has a coke relatively high in organic sulfur been tried in a blast furnace or cupola to learn whether there is any actual difference in practice?

A. St. John.—In the coking process, the sulfur present as organic sulfur is liberated and exists in the coke as some inorganic form of sulfur. By means of X-ray crystal analysis we can show in the solid formation the combination of atoms constituting a great many materials that are existing in a crystalline form, that is, having an orderly arrangement of atoms, although the actual particles are so small that you cannot see them in a powerful microscope. There are many cases where it is utterly impossible by chemical analysis to tell whether you have one state of combination of a group of atoms or another; X-ray analysis discloses this beyond a doubt.

I have recently succeeded in extending the sensitivity of these analyses to the point where I could determine 0.2 per cent. of a minor constituent in a mechanical mixture of two constituents. It is now possible to have these analyses carried on in a consulting laboratory.

R. H. Sweetser, Columbus, O.—The Portsmouth By-Product Coke Co. had a coal going into the coke ovens that had considerable organic sulfur, so reported by the chemist. It was greater in amount than the

[29] A. R. Powell: Forms of Sulfur in Coke: *Jnl.* Am. Chem. Soc. (1923) **45**, 1–15.

Some Factors Affecting the Sulfur Content of Coke and Gas in the Carbonization of Coal. *Ind. & Eng. Chem.*, (1921) **13**, 33–35.

Quantitative Determination of Sulfur Forms in Coke, *Ind. & Eng. Chem.*, (1923) **15**, 951–53.

Analysis of Sulfur Forms in Coal, Bur. of Mines *Tech.* Paper 254 (1921) 21.

[30] A. R. Powell: Forms of Sulfur in Coke, and Their Relations to Blast-furnace Reactions. *Trans.* (1923) **69**, 587–99.

pyritic sulfur. We were surprised to find that in the coke most of the sulfur was reported as organic sulfur. One of our fellow members, C. B. Murray, made the analysis.

Any kind of sulfur is objectionable in the blast furnace. It does not blow out of the top but goes immediately into the iron sponge, and is not taken out until it reaches the tuyere zone.

T. DeVenny, Edgarton, W. Va.—In the investigations of the various low-sulfur coals—that is, coals carrying from 0.50 to 0.60 per cent. sulfur—in almost every instance was not the greater part of the sulfur organic? In other words, organic sulfur in coal very seldom goes below 0.40 per cent.

A. C. Fieldner.—Yes; in coals containing from 0.5 to 1.0 per cent. sulfur, the organic sulfur varies between 0.4 and 0.7 per cent. The following table[31] gives sulfur forms analyses of some typical coals:

Summary of all Analyses for Sulfur Forms in Coal

Sulfur Forms	Coal Samples					
	No. 23066 (Pa.) Per Cent.	No. 18847 (Pa.) Per Cent.	No. 20507 (W. Va.) Per Cent.	No. 21308 (Ky.) Per Cent.	No. 21100 (Tenn.) Per Cent.	No. 27224 (Kans.) Per Cent.
Pyritic sulfur	0.47	0.79	0.08	0.13	1.75	1.99
Sulfate sulfur	0.07	0.23	0.01	0.04	0.71	0.32
Humus organic sulfur	0.50	0.46	0.44	0.42	1.01	0.39
Phenol soluble organic sulfur	0.12	0.20	0.02	0.09	0.77	0.32
Total	1.16	1.68	0.55	0.68	4.24	3.02
Total by direct analysis	1.21	1.72	0.56	0.71	4.25	3.06
Difference between totals	0.05	0.04	0.01	0.03	0.01	0.04

T. DeVenny.—Then, in low sulfur coal you can assume that at all times you can find at least 0.40 per cent. organic sulfur; and as in the coking process, we do not find that organic sulfur passes off as a gas, it would be hard to expect a coke that would go below 0.40 per cent. sulfur.

S. A. Taylor, Pittsburgh, Pa.—In an X-ray analyses do you get a spectrum that determines the amount or location of sulfur?

A. St. John.—It depends on what we are trying to find. In the case I first mentioned, we took simply a shadow view of a lump of coal, the same shadow that the doctor takes when he thinks a patient has a frac-

[31] From U. S. Bur. of Mines *Tech. Paper* 254, p. 14.

tured rib, and we saw the darker shadows due to the ash material. In the analysis for intrinsic ash and removable ash, the method is similar.

The finely divided sample is placed in a container of a certain definite thickness, say 1 in., jolted or "jigged" in a definite way and then exposed before the X-rays. It is found that a lot of particles containing high atomic weight material, and particularly individual ash particles go to the bottom, giving a dense shadow in the bottom. The relative position of the line of demarkation, which is fairly definite between the high absorption at the bottom and the lower absorption at the top, checks very closely with the flattened point on the Henry curve.

The study of the physical condition in which the constituents are present is effected by diffraction analysis, whereby you get a pattern of lines on a photographic film, the positions of the lines being characteristic of the materials that are present in the sample, and it is from an analysis of the positions and the relative intensities of those lines that we arrive at a knowledge of the components and the percentages in which they are present.

A. C. Fieldner.—Two years ago I visited Professor Kemp and Mr. McLaren in their laboratory at Edinburgh, where they had been applying X-ray examination to coal. They had a stereoptic arrangement for viewing a lump of coal under the X-ray, which made the lump look like a chunk of ice showing the bands of slate, particles of pyrite, and other impurities in the coal as has just been described. The particles of pyrite appeared like opaque bodies frozen into the ice. A lump of coke made from crushed coal likewise showed little pieces of slate and ash-forming material scattered all through it. I think the X-ray will prove to be useful in coal-washing problems.

Relation of Origin and State of Carbonization of Coal to Problems of Low-temperature Carbonization

By S. W. Parr,* Urbana, Ill.

(New York Meeting, February, 1926)

The extent to which geological carbonization has taken place in the process of coal formation is a fundamental factor in all considerations relating to classification, oxidation, deterioration, spontaneous combustion, and especially in all considerations relating to destructive distillation having as an objective the production of marketable coke and by-products. Moreover, it may be readily shown that a striking relationship exists between the geological decomposition or geological carbonization processes and those artificially produced by heat. This may be well illustrated by bringing again to view some of the charts more commonly met with 25 years ago, as in Fig. 1.[1]

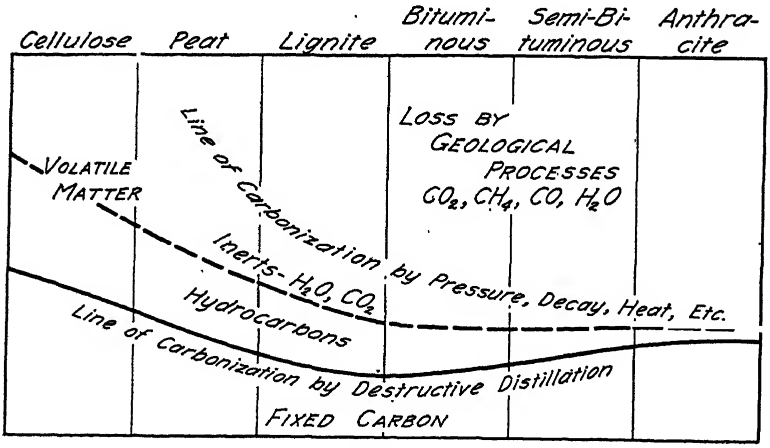

Fig. 1.

In this figure the line of geological decomposition by decay, pressure, etc., because of its relation to the line of carbonization by destructive distillation, becomes of special significance when we attempt to formulate any system of classification which shall be reasonably scientific and at

* Professor of applied chemistry, University of Illinois.

[1] S. W. Parr: Composition and Character of Illinois Coal. Illinois State Geol. Surv. *Bull.* No. 3 (1906) 29. Fig. 1 adapted from article cited.

the same time of practical or industrial value. We are all familiar, for example, with the method of designating the coking quality of a coal by reference to the amount of residual oxygen in the coal substance—or the ratio of such oxygen to the hydrogen present. This suggests the possibility of devising a method of coal classification which shall at once indicate the degree of geological carbonization and the possibility of artificial carbonization, by destructive distillation.

Such a method seems to be met in a fairly satisfactory manner by a system of classification in which the oxygen present makes its presence evident by its negative effect on the calorific value of the pure or unit coal substance. The application of this principle is shown in Fig. 2.[2]

Types of Coal

It is not the purpose here to present a plan for coal classification, but the outline as presented in Fig. 2, has this in its favor, that the boundaries which set off the various types, being the combined result of calorific values and content of volatile matter, define in a positive manner the well-known coking coals. For example, the zones *A* and *B* represent the low and high volatile coals, respectively. These are the so-called coking coals, while zone *C* including the mid-continental coal areas, and zone *D*, the coals of the sub-bituminous type, are commonly designated as non-coking in character. The chart, therefore, is a visual representation of the relative extent to which geological carbonization has taken place. It is to be noted that high carbonization is accompanied by deoxygenation and the increase in carbonization as we advance from zone *A* to zone *B* is accompanied by an augmented oxygen content, as shown by calorific values, which are progressively lowered until the limiting line of 15,000 is reached at the border line of zone *C*.

The facts thus presented, in general outline, bring us directly to the question as to what constitutes the coking property and what are the transformations, either physical or chemical, by which it is accomplished.

We use the term "carbonization" but it has a somewhat too restricted meaning to quite answer our purpose in a discussion of this sort. For example, we can carbonize sawdust, but without producing charcoal. Or we may carbonize lignite and not produce coke, or we may carbonize finely divided coal from the Pittsburgh seam and obtain a good

[2] From an unpublished thesis by Elmer Vliet, Univ. of Illinois, 1926.

Definition for "unit coal" and method for calculating its heat value has been set forth elsewhere; see Parr and Wheeler: Unit Coal and the Composition of Coal Ash. Univ. of Illinois Eng. Exp. Sta. *Bull.* No. 37. Formula for unit B.t.u. is as follows:

$$\text{B.t.u. (unit coal)} = \frac{\text{B. t. u. as received} - 5000S}{1 - (\text{moisture} + 1.08\,\text{ash} + .55S)}$$ and per cent. volatile matter (unit coal) $$= \frac{\text{Vol. (as rec.)} - (.08A + .4S)}{1 - \text{moisture} + (1.08A + .55S)}.$$

quality of coke. Hence, it would seem that, for purposes of discussion at least, the word carbonization may properly carry with it the idea of destructive distillation under conditions which produce a coherent mass of material which we designate as coke. This also is the idea involved when we speak of a coking or a non-coking coal. With this added feature of coherence the topic at once becomes complicated under

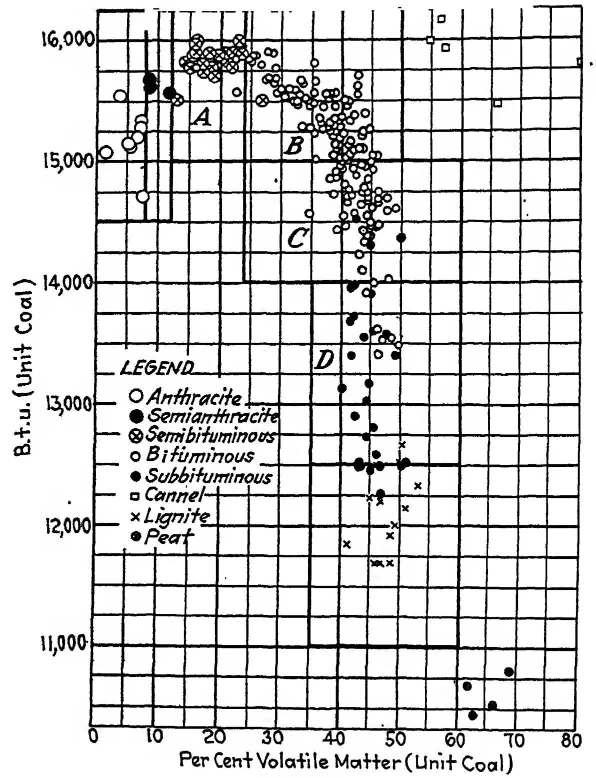

Fig. 2.

conditions of artificial carbonization by means of heat. Under geological conditions, however, we may have a minimum of heat but a maximum of pressure resulting in a good degree of coherence, but the coherence thus secured is not an index of the extent to which the geological carbonization process has been carried, nor does it carry any suggestion as to what may be expected, when artificially carbonized.

Conditions of Carbonization

To bring about the double effect of carbonization and coherence, or of the broader significance of the term "carbonization" with the formation

of coke, the essential conditions are by no means simple, and perhaps a warning should be here noted that the full information is not always revealed by the single factor of unit heat value.

Melting Point

In order to have a coherent mass in the finished product there must be sufficient material present with a relatively low melting point to bring about at some stage of the process a pasty or semi-liquid condition throughout the mass of material. Such fusible material may be of two general types, (*a*) resins, and (*b*) bitumens. The resins of the original plant material have undergone relatively little alteration throughout geological changes. They may, on the application of heat, volatilize to a considerable extent without much change in composition. The residue, therefore, upon complete carbonization, is small. These features, coupled with the relatively small amount of resins present in most plant structure, will explain why the substances of the resinic type are of negligible importance in furnishing the agglutinating or cementing property in the process of carbonization. Illustrations of this feature will readily present themselves. The cannels which are high in resins, and, it should be noted, high in unit heat value, have little or no coherence upon carbonization. The other type of fusible material (*b*) is designated as bituminic in character. It is here that our interest centers for the reason that this substance is a secondary or decomposition product of the original vegetal matter and, therefore, of decided interest as a geological product; but it is of equal or greater interest in its behavior under carbonization conditions because potentially it carries the true coking or agglutinating property of the coal when subjected to carbonization conditions.

Bituminic Material

It is here that we come to the crux of the matter. As we pass from zone *B* into zone *C*, the question arises as to whether the deficiencies in coking property are due to a lack of bituminic material or to its different quality, and especially in what way does the high oxygen content affect the carbonization process.

In the large coal area of Illinois, Indiana, and Western Kentucky, the splint or non-coking property is rarely met with, and from even a general acquaintance with practically all of these coals the coking or bituminic property can be readily established. Hence, the great interest that attaches to studies on this type, particularly as to the character of their bituminic substance.

Oxidation Effect

Briefly, without going into the detail of experimental data, it has been shown that by adding to the oxygen content, as in weathering or artificially satisfying the very marked avidity of these coals for oxygen, the coking property may be entirely destroyed. Again, if by means of a suitable solvent we separate the bituminic material from the lignitic residue and saturate this residue with oxygen, then recombine the insoluble with the soluble portion in their original proportions we will find upon heating that the coking property has been destroyed. After an elaboration of experiments along the line of the effect of an increase or a diminution of oxygen in the two type substances as obtained by solvent separation, or studies in the solvent property of the bituminic substance upon the lignitic material, or the effect of inter-reactions of a chemical nature between the two or between their products of decomposition at elevated temperatures, or again, studies of the changes or even the total obliteration of a melting point for the bituminic matter by varying the rate of heating—the fact is revealed that the bituminic material of these coals upon which the coking property depends is of a highly sensitive character, chemically speaking, and that the successful carbonization of this type of coal, as it comes to our hand from geological process, must take into consideration these rather involved and more or less obscure properties. We may summarize the fundamental conditions to be observed as follows: (*a*) the avoidance of oxygenation; (*b*) the lowering, so far as may be possible, of the oxygen content of the coal; (*c*) the removal of the initial gaseous products of decomposition, such as CO_2 and H_2O, which occur before the pasty or carbonaceous stage of decomposition has been reached; and (*d*) the quick approach to the softening temperature.

Low-Temperature Carbonization

A word now as to the high-temperature or low-temperature method of carbonization as applied to coals of the *C* type. In the process of carbonization, all bituminous coals deliver first a gas which is heavy, largely condensable and of the so-called rich type, as contrasted to the gases that follow, which are lighter or leaner. The coals of the *B* zone deliver their lean gases readily, immediately following the discharge of their rich gases and without any great expenditure of additional heat or time. In contrast to this the coals of the *C* zone deliver their rich gas even more readily and of even a heavier or richer quality, while the lean gases are held more tenaciously and are at the same time of a leaner or poorer quality than the lean gases of type *B*.

It is here that the chief argument for the use of lower temperatures in the case of *C* coals becomes apparent. These light products are

worth more in the coke than they are in the gaseous products. We may therefore save in the matter of both time and heat if we can effect the carbonizing process within the so-called low temperature range.

The problem is therefore definable in what would seem to be fairly simple terms. Its accomplishment seems to be not so readily attained if we may judge from the vast expenditure of funds and time and the elaboration of apparatus and equipment in the effort toward its solution. Industrially it is still a goal ahead to be striven for and it is evident to anyone who attemps to follow the increasing number of publications on the subject that the methods vary with regional requirements and with the sort of material to be worked, but in all the wide range of material from the true coking coals through the various lignites, the logic of the situation throughout favors the low-temperature idea and the purpose of this paper will be accomplished if it has been shown that there is nothing inherently impossible in its solution.

DISCUSSION

A. C. Fieldner, Pittsburgh, Pa.—This paper reminded me of Thiessen's work on the banded structure of coal—that is, the dull and bright bands of bituminous coal. He separated some of these dull and bright bands with the aid of a microscope and subjected them to the coking process. As you know, the dull bands are composed of the attritus and fine débris of spore coats, pollen grains, waxy coverings, and similar materials of plant remains, whereas the bright bands, which he terms "anthraxylon," are the pieces of coalified wood and show characteristic woody structure.

In coking these two bands there was evident a tendency to fuse in the anthraxylon or bright material, and a lesser one in the attritus or dull bands, which bears out the author's statement that the cannel coals, which are composed largely of waxy spore coats and resins, do not melt and coke. The coking properties seem to be in the humic portions of the plant remains rather than in the waxes and resins.

Coking is associated with certain limits of percentages of oxygen. Lignite, with high oxygen content, does not fuse or coke, and anthracite, with a low oxygen percentage, does not fuse. Between certain oxygen limits, coals may fuse and coke. A coking coal can be slightly oxidized artificially and its coking properties destroyed. Certain constituents can be extracted from coking coal with solvents which leave a non-coking residue, neither does the extracted material by itself make a good coke. But on mixing the extracted material with the residue the mixture again becomes coking. A high resin and wax content does not necessarily impart coking properties to a coal, as shown by the non-coking nature of certain Utah coals. These coals go through the coking process without

any material fusion. The lumps come out of the retort in the same shape as they were charged and are only shrunken and somewhat checked, and yet this coal has a large proportion of material that can be extracted by solvents. It is full of resins. Lumps of resin may be seen in the coal, and the resin may be distilled out of the coal with steam. The distilled resins are not so brittle as the original resins, but are still soluble in the same solvents. It is obvious that the coking properties do not reside in these resinic, waxy materials, but rather in some other degradation product or combination of products.

The Pittsburgh Coal Bed

By I. C. White,* Morgantown, W. Va., G. H. Ashley,† Harrisburg, Pa., and J. A. Bownocker,‡ Columbus, Ohio

(Pittsburgh Meeting, October, 1926)

Among the rich mineral deposits of the great Appalachian field, the Pittsburgh coal bed stands pre-eminent. Other coal beds may cover a wider area, or extend with greater persistence, but none surpasses the Pittsburgh seam in economic importance and value. It was well named by Rogers, and his able assistants of the First Geological Survey of Pennsylvania, in honor of the city to whose industrial growth and supremacy it has contributed so much. Whether or not the prophetic eye of that able geologist ever comprehended fully the part which this coal bed was to play in the future history of the city that gave it a name we do not know; but certain it is that the 7 ft. of fossil fuel, which, in Rogers' time, circled in a long black band around the hills and, overlooking the site of Pittsburgh from an elevation of 400 ft. above the water of the Allegheny and Monongahela, extended up the latter stream in an unbroken sheet for a distance of 200 miles, has been the most potent factor in that wonderful modern growth that has made the Pittsburgh district the manufacturing center of America, and that bids fair to continue until it shall surpass every other district in the world, even if it does not now hold such primacy.[1]

The Pittsburgh coal bed stands today as probably the largest contributor of wealth of any single mineral deposit in the world. If it is not, what other deposit is? It has contributed to Pennsylvania alone more than 2,500,000,000 tons of high-grade coal worth nearly $4,000,000,000; since 1910, it has added more than $2,500,000,000 to the value of Pennsylvania's mineral output. In 1920, this bed[2] contributed 127,227,000 tons of coal as follows:

Pennsylvania, Pittsburgh district	38,245,000	
Connellsville district	34,095,000	
Westmoreland-Ligonier district	16,107,000	88,447,000
West Virginia, Fairmont district	15,571,000	
West Virginia-Ohio, Panhandle No. 8 district	21,128,000	
Maryland-Potomac (Est. one-third of production)	2,081,000	38,780,000
Total		127,227,000

* State Geologist of West Virginia.

† State Geologist of Pennsylvania.

‡ State Geologist of Ohio.

[1] I. C. White: *Proc.* Amer. Ad. Sci. (1897) 187.

[2] U. S. Geol. Surv., *Min. Resources of the U. S.*. 1923, Part II (1926) 534.

Assuming that West Virginia, Ohio, and Maryland have contributed in about the same ratio during past years, it may be estimated that the total production from the Pittsburgh bed has been not far from 3,500,000,000 tons, and probably there is still in the ground several times this quantity.

It would be difficult to visualize the large part coal from the Pittsburgh bed has played in the economic development of the United States. The Potomac, with the Chesapeake and Potomac canal, has acted as an outlet for the Maryland-Pittsburgh bed; the Youghiogheny, Monongahela, and Ohio Rivers afforded cheap transportation for the coal of West Virginia, Pennsylvania, and Ohio, to the broad Mississippi Valley, both northern and southern; the Great Lakes afforded cheap transportation to central Canada and to the far northwest, and the Erie Canal to central and eastern New York and New England. In the early days, canals and a canal railroad over the mountains gave water transportation to eastern Pennsylvania and Maryland and the New York market. Pittsburgh coal early became noted for its excellent qualities as a gas and coke-making coal, and as a strong steam coal. In the early days, gas coal from Westmoreland County was used from the Atlantic seaboard to Chicago in making artificial gas. Coke, especially that from the Connellsville district, served blast furnaces through much of the eastern United States so that, quite aside from serving as a foundation for the great iron and steel industry of the Pittsburgh district, this coal bed has served the wants of half a nation or more.

Present Extent

At present, the Pittsburgh coal bed is confined to the northern part and the eastern and western edges of a major syncline that extends southwestward from Pittsburgh and centers about at the southwest corner of Pennsylvania (Fig. 1), and to detached remnants lying far east and north of the main body of coal. As known, at present the Pittsburgh bed is lacking under a large area in the southwest part of its basin, an area in which it was long supposed to exist. According to the best information now available, the area of the Pittsburgh coal present when mining began was as follows:

Area of Pittsburgh Coal	Square Miles
Pennsylvania	2077
West Virginia	2214
Ohio	1410
Maryland	28
Total	5729

Aside from the large area in West Virginia and Ohio where the coal is lacking under cover, the presence or absence of the bed at any place

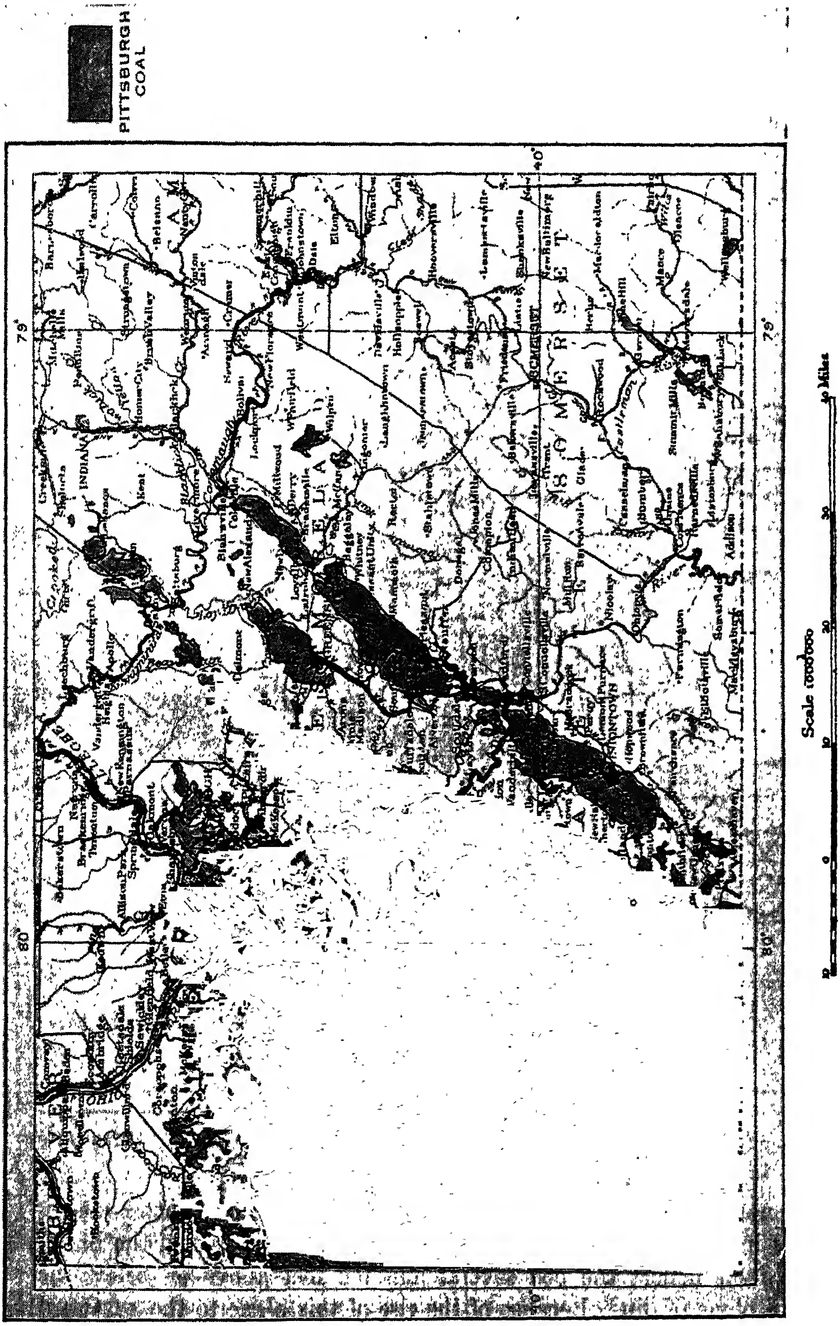

FIG. 1.—MAP SHOWING AREA UNDERLAIN BY PITTSBURGH COAL.

has been determined by the structure given the bed at the end of the Carboniferous Age, and by the subsequent erosion. The main body of the bed is found in the major syncline lying between the Appalachian Mountains on the east and the Cincinnati arch on the west. The western flank of this basin is a slightly rolling slope dipping, in general, south and east toward the center of the basin. The eastern flank has a general dip

FIG. 2.—MAP SHOWING LINE OF OUTCROP OF PITTSBURGH COAL IN OHIO, AND LOCATION OF THE THREE PRINCIPAL FIELDS.

to the northwest toward the axis of the basin, but is thrown into a succession of irregular folds, in general, parallel to the main axis of the basin. These folds deflect the bed several hundred feet above or below the general plane of dip. Because of the rise of this plane to the eastward and the spooning out of the major syncline to the northward, the coal is carried to higher and higher levels, going eastward or northward, so that

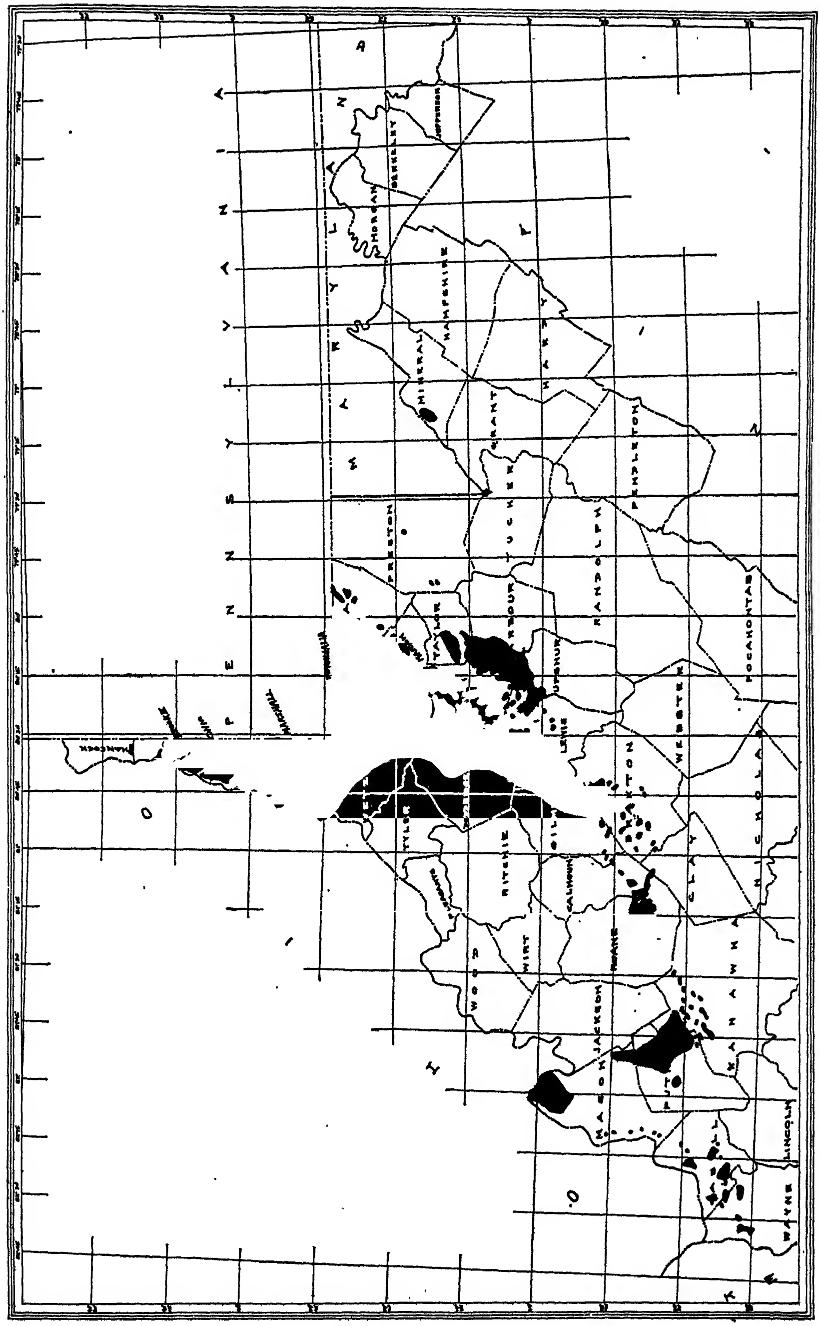

FIG. 3.—OUTLINE MAP OF WEST VIRGINIA SHOWING PITTSBURGH COAL (SHADED AREAS). FROM MAPS OF THE WEST VIRGINIA GEOLOGICAL SURVEY. SCALE: 1 IN. = 48 M. APPROXIMATELY.

beyond the center of Westmoreland and Fayette counties, Pa., and Monongalia, Marion, Harrison, and western Lewis and Braxton counties, W. Va., the tops of the arches have been removed and the coal is found only in the bottoms of the basins where it has been preserved from erosion. Such basins exist in Armstrong, Indiana, eastern Westmoreland, and Fayette counties, Pa., and in eastern Monongalia, Marion, Harrison, Preston, Taylor, Barbour, Upshur, Grant and Mineral counties, W. Va., Allegany County, Md., and in eastern Somerset County, Pa., nearly 50 miles east of the main body of the bed.

As shown in Figs. 2 and 3, the workable coal ends abruptly along a line through northern Monroe County, Ohio, and Wetzel, Doddridge, and Gilmer counties, W. Va. The reason for this disappearance of the Pittsburgh coal, as revealed by the drill of the petroleum and natural gas operators, has always been attributed by the senior author to deeper water conditions prevailing westward from the old Pittsburgh peat swamp in which the bog vegetation of the time could not grow and accumulate deposits of vegetable material. This view is sustained by the presence of black shale, apparently a water deposit, at the horizon of the Pittsburgh bed. The map also reveals the patchy nature of the Pittsburgh coal areas southwest from Braxton County.

Original Extent

The isolated areas of Pittsburgh coal in deep synclinal basins in Somerset County, Pa., Allegany County, Md., and Preston, Tucker, Grant, and Mineral Counties, W. Va., render the conclusion practically certain that this wonderful coal deposit once covered a belt at least 50 miles wide east of its present principal body, and may have extended much farther eastward. Also detached areas in Jefferson County, Ohio, Beaver, northern Allegheny, Armstrong, and Indiana counties, Pa., indicate that the bed may have originally covered an area far north of its present northern boundary. As early as the First Geological Survey of Pennsylvania, J. P. Lesley, one of the assistants of that Survey and chief of the Second Geological Survey, believed that this great coal deposit may once have extended in an unbroken sheet eastward to the Blue Ridge in eastern Pennsylvania and the eastern boundary of West Virginia. However this may be, undoubtedly an enormous area of this fuel has been removed by erosion from western Pennsylvania, western Maryland, northeastern West Virginia, and eastern Ohio. It was formerly thought that thin coals in eastern Cambria County and in the Broad Top field of Pennsylvania represented this bed. Recent drilling and studies have shown that in northern Somerset County the interval from the Upper Freeport coal to the Pittsburgh coal has increased from the usual 600 to about 900 ft. and suggests that the Pittsburgh horizon is not caught by the hills in either eastern Cambria County or in the Broad Top field.

The Tracy coal bed of the anthracite field has been thought, by David White, to be about at the horizon of the Pittsburgh bed; if this bed is an extension of the Pittsburgh bed, the original extent of the Pittsburgh bed to cover the anthracite region is indicated.

Variation of Character

Naturally a coal bed underlying so wide an area as this varies much in character. In Somerset County, Pa., Allegany County, Md., Mineral and Grant counties, W. Va., this is a low volatile, semi-bituminous, semi-smokeless coal of high grade having 18–24 per cent. volatile matter, and 63–70 per cent. fixed carbon. Farther west, the percentage of volatile matter increases and the coal is a high-grade coking coal having 25–33 per cent. volatile matter, and 55–63 per cent. fixed carbon. Still farther west, the volatile matter increases slightly, the fixed carbon drops, and the coal becomes a "gas coal," typified by the coal of the Greensburg basin and lower Youghiogheny region. Around Pittsburgh and to the west and southwest, this bed is a strong steam coal, showing, however, an increased percentage of sulfur. Physically, the coal changes from a tender coal at the east, producing so little lump coal that it is largely marketed as run-of-mine, to a strong "block" coal, in the western counties especially north of Brownsville, producing a large percentage of lump coal. Also, the thickness changes from 22 ft., in the Elk Garden basin of Mineral County, W. Va., to 14 ft., or more in the Georges Creek basin, Md., to 5–9 ft., in southwest Pennsylvania, to 10–11 ft., in Preston and Tucker counties, W. Va., and 4–7 ft. in the more westerly counties of West Virginia, and an average of 4–8 ft. in Belmont County, Ohio, and becomes thinner to the southwest.

Type Section

It was early discovered by Stevenson, an assistant geologist of the Second Geological Survey of Pennsylvania, that over a very large area in southwest Pennsylvania, in the Belmont field of Ohio, and in northern West Virginia, the Pittsburgh bed maintains a remarkably uniform bed section, as follows:

Type Section of Pittsburgh Coal Bed

1. Roof coals, 0–8 ft. thick, usually broken up with clay or shale partings so as to be unminable, though locally solid, or nearly so, and mined.
2. Over, or main, clay, 3 in., 3 ft. thick, usually 6–8 in., locally other beds come in, so as to separate the roof coals and the coals below by as much as 20 ft.
3. Breast coal, averages about 4 ft. of good clean coal.
4. Parting, shale or clay, ¼ to ¾ in.
5. Bearing-in coal, 2–8 in. thick, mined in days of pick mining.
6. Parting, shale or clay, ¼ to ¾ in.
7. Brick coal, a few inches to 3 ft.; mines out in brick-shaped blocks.
8. Parting of shale, often lacking.
9. Bottom coal. About 1 ft. thick, often impure and frequently left in mine in Pennsylvania.

Of this section, part 1 is often called the "roof division;" and parts 3 to 9, the "main division." In nearly all places in Pennsylvania, where this type prevails, the good commercial coal is confined to the "breast" and "brick" subdivisions. The "roof" and "bottom" coals were formerly left in the mine as too impure (high in ash and sulfur) to warrant mining. It was a great waste and, sometime in the future, when the thick coals of the Appalachian field are approaching comparative exhaustion many areas of this bed, especially within 30 to 40 m. of Pittsburgh, will probably be reworked for this coal.

Pittsburgh Bed in Pennsylvania

While the section given is typical of a large area in Pennsylvania, Ohio, and West Virginia, none of these features is persistent over the

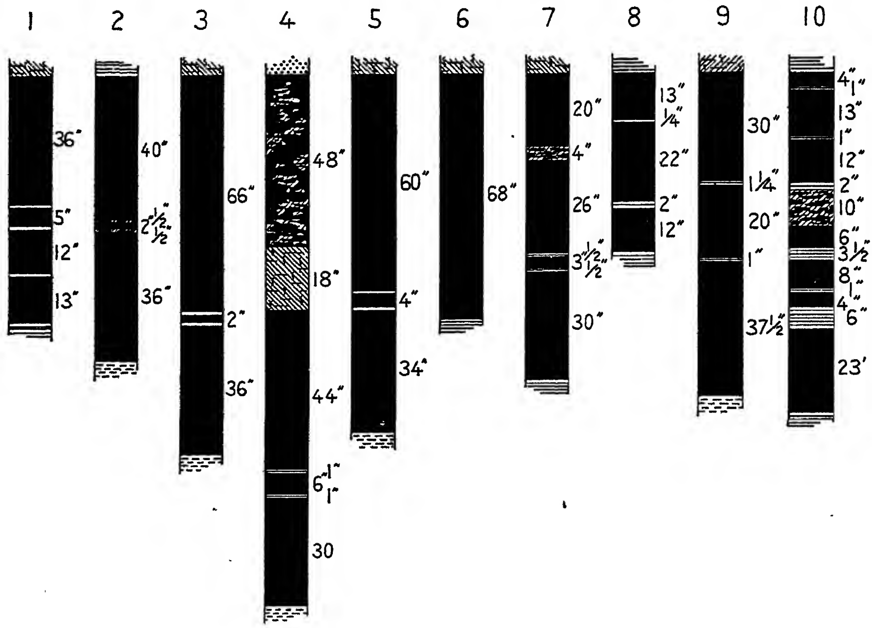

Fig. 4.—Typical sections of Pittsburgh coal bed in Pennsylvania.

1. Allegheny County.
2. Washington County.
3. Connellsville basin, Fayette County.
4. Greensburg basin, Westmoreland County (whole bed).
5. Latrobe basin, Westmoreland County.
6. Indiana County.
7. Ligonier Valley, Westmoreland County.
8. Pine Hill basin, Somerset County.
9. Salisbury basin, Somerset County.
10. Wellersburg basin, Somerset County.

whole basin. Parting shale No. 8 is widely lacking. Partings Nos. 4 and 6 disappear in sections of the field and other partings appear. Fig. 4 shows sections, selected from hundreds, that are fairly typical of the areas of this bed in Pennsylvania. All the sections, except section 4,

are of the main division only. As the partings in any area become irregular, no one section can be selected as typical.

In general, it may be said that the Pittsburgh coal is approaching exhaustion in the Wellersburg, Salisbury, Pine Hill, and Ligonier basins. The coal of the Latrobe, Greensburg, and Connellsville basins will be exhausted within 30 years, if the past rate of production is maintained.

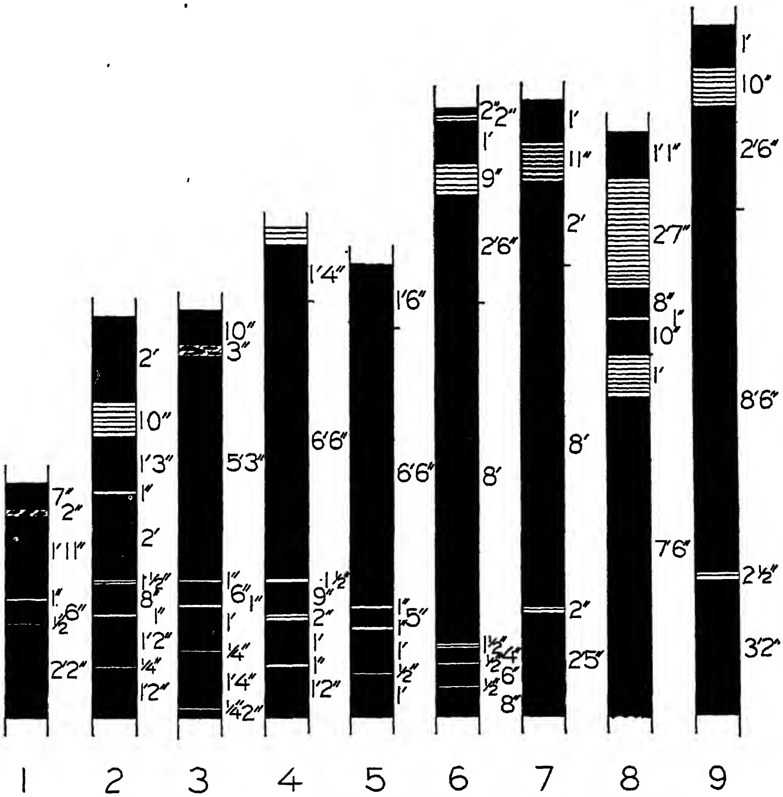

FIG. 5.—SECTION OF PITTSBURGH COAL BED IN MARYLAND.

1. Bald Knob, north of Mt. Savage.
2. North of Allegany.
3. At Frostburg.
4. At Midlothian.
5. At Carlos.
6. Near Lonaconing.
7. Near Bartlett.
8. Near Phoenix.
9. Franklin Hill, near Westernport.

Western Westmoreland and Fayette counties still have considerable bodies of coal. Pennsylvania's great body of unmined coal, however, lies in Greene and Washington Counties, which are almost wholly underlain by the Pittsburgh bed. In Greene County, mining on this bed has only started; and in Washington County the bulk of the coal is as yet untouched. The coal thickens from north to south, approaching 5 ft. around Pittsburgh and 9 ft. in Greene County.

Pittsburgh Bed in Maryland

In Maryland, the Pittsburgh bed is confined to a deep, broad syncline crossing the state from north to south mostly in Allegany County. The syncline lies between the Dans—Allegheny Mount. belt on the east, and the Big Savage Mountain belt on the west,—an area of about 20 by 5 m. The Pittsburgh bed, long known as the "Big Vein," is confined principally to the northern end of this basin; because of its high position in the stratigraphic column, in the south part of the basin, it occurs only in isolated hilltops.

Fig. 5 shows the character of the bed in Maryland.

The first mining of the Pittsburgh bed in the Georges Creek basin is nearly finished; undoubtedly, second mining will obtain the coal left in first mining.

Pittsburgh Bed in West Virginia

Passing up the Monongahela River, into West Virginia, considerable change takes place in the structure of the Pittsburgh coal bed. The shaly partings are still there 4 to 5 ft. above the under-clay, but the brick and bottom members have united into one mass 4 to 5 ft. thick, which becomes the important part of the bed. The breast bench has 1 ft. or more of bony coal at its top, which is generally utilized as a roof support for the over-clay, so that only about two-thirds of the 3 to 3½ ft. of breast coal is mined as good fuel. The shaly partings are often three in number and the bearing-in coal, with its partings, attains a thickness of 7 to 8 in. This is the type in Monongalia County, Marion, and much of Harrison. Farther south in Lewis, Gilmer, Braxton, Roane, and Kanawha counties, only one band of shale may be present, about 4 ft. above the under-clay, or near the middle of the bed; and when the coal gets thin, it is usually the portion above the shale band that disappears first.

In Mason, Putnam, and Cabell counties, a coal bed in the Monongahela series (the same one mined at Pomeroy, Ohio) that was formerly supposed to represent the Pittsburgh bed has been correlated by Bownocker, State Geologist of Ohio, with the Redstone coal, which belongs 40 to 50 ft. above the Pittsburgh bed. The evidence for this correlation rests largely on the presence of a yellowish limestone a few feet below the coal bed in question. But as a yellowish limestone is often found a few feet below the genuine Pittsburgh coal bed, C. E. Krebs, who as Assistant Geologist of the West Virginia Geological Survey, collected the data on Jackson, Mason, and Putnam counties, is not convinced that the Pomeroy or Mason City coal is the Redstone instead of the Pittsburgh; it has hence been carried under the latter designation in the West Virginia publications. If it is the Redstone, the Pittsburgh bed is entirely absent from that region of the state, for only one workable coal bed is present at this horizon. (Fig. 6.)

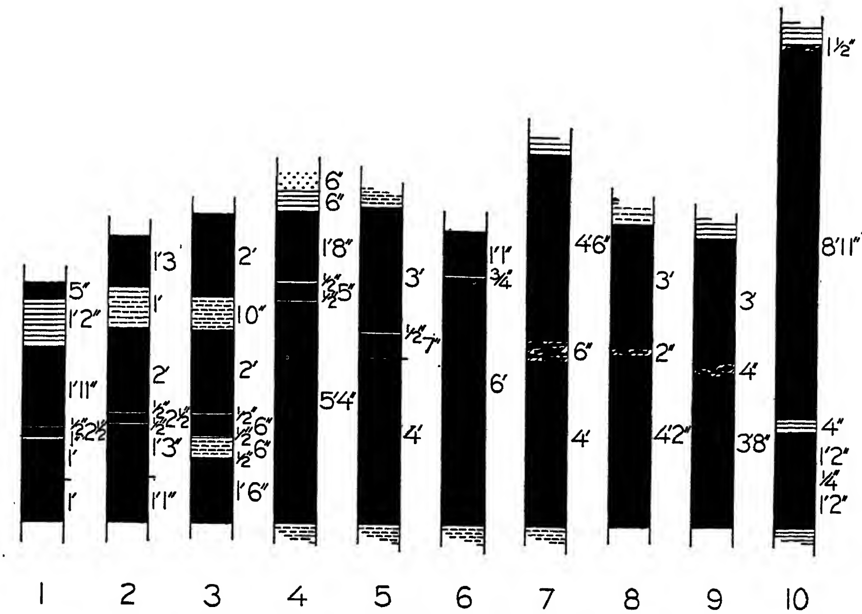

FIG. 6.—SECTIONS OF PITTSBURGH COAL IN WEST VIRGINIA.

1. Brooke County.
2. Ohio County.
3. Marshall County.
4. Monongalia County.
5. Marion County.
6. Harrison County.
7. Barbour County.
8. Gilmer County.
9. Braxton County.
10. Mineral County.

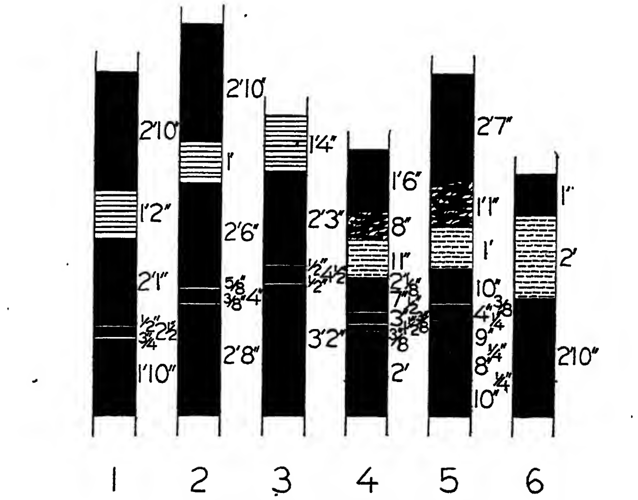

FIG. 7.—SECTIONS OF PITTSBURGH COAL IN OHIO.

1. Harrison County.
2. Jefferson County.
3. Belmont County.
4. Morgan County.
5. Athens County.
6. Gallia County.

Fig. 6 shows the character of the Pittsburgh bed in the several counties of West Virginia.

Pittsburgh Bed in Ohio

The structure of the Pittsburgh bed in Ohio is shown in Fig. 7. On account of the more irregular development, its thickness and character are described more in detail than in the other States. Belmont field, the most important field of Pittsburgh coal in Ohio, is in Belmont County, lapping over into Jefferson, Harrison, Guernsey, Noble, and Monroe counties. Within its outcrop, the Pittsburgh bed in this field appears to be everywhere present except in the valley of the Ohio River and in two or three narrow tributary valleys. In the southern two-fifths of Belmont County, the Pittsburgh coal has scarcely been touched and it constitutes the largest known block of undeveloped coal in Ohio. The average thickness of the bed, exclusive of partings, in Belmont County, based on measurements in thirteen mines, is 4 ft. 10 in. The maximum, exclusive of roof coal, is reported as 8 ft. Along the western margin, it measures about 4 ft. and decreases in a very short distance to only 1 ft. In Jefferson and Harrison counties, the coal is confined to the ridges and hills and, because of its accessibility, has been worked extensively for 40 years. The thickness is similar to that of Belmont County, averaging about 4 ft. 5 in. While the coal in the Belmont field varies from place to place, it is marked by uniformity of thickness and composition and regularity in structure. Although occasional anticlines or folds are found, the rocks on the whole have a fairly uniform dip, which Stout reports as south-of-east 20 ft. per mile. The roof of the coal in Belmont County is a weak clay shale, except in the western part where it is sandstone. Roof troubles are common and are said to increase the cost of mining from 30 to 50 cents per ton. Farther north, in Jefferson and Harrison counties, the roof coal, in places 2 ft. thick, is commonly present and is left as a support. Clay veins, from a few inches to 20 ft. wide, are common.

Between the Belmont and Federal Creek fields, the Pittsburgh coal is thin and unimportant. It is found near the hilltops in southeast Muskingum County, where it is from 18 to 30 in. thick. The coal is of even less importance in Noble County, except on ridges that form an extension of the Belmont field. Much the same is true in Morgan County, east of Muskingum River, where the coal is as much as 30 in. thick, but in most places it is only 15 in. West of the Muskingum, the coal does not appear to be important until the Federal Creek field is reached.

The Federal Creek field, of the Pittsburgh coal, includes parts of Homer and Marion townships, in the southwest part of Morgan County, and Ames and Bern townships, in the northeast part of Athens County. The area is drained by Federal Creek and its tributaries, hence the name of the field.

So far as Morgan County is concerned, the Pittsburgh coal is at its best in Homer township, but as the bed lies near the summits of ridges and hills, the area is small. In Marion township, which lies east of Homer the coal is of value in the southwest corner only and the area probably does not exceed 4 square miles.

The area of workable Pittsburgh coal in Athens County, although larger than in Morgan, is by no means great. In Ames township the coal lies well up in the ridges and hence the area is small. Bern township lies east of Ames and, because of the dip, the coal is lower and the area therefore is larger. Along the eastern border of this township, the coal is below drainage. To the south, in the valley of Federal Creek, the coal falls below drainage in section 18, Rome township. As indicated in section 5, the coal is in two benches, separated by a layer of clay about 1 ft. thick. The thickness varies greatly within short distances, due largely to the thinning or complete absence of the upper bench. In places the upper bench—much less commonly the lower bench—carries bowlders of sandstone or of sandstone and pyrite, which of course decrease the value of the coal. The bowlders differ greatly in size and shape; the largest weigh a ton or more, and some are round, some oval, and some flat. Probably they were formed from sand deposited on the vegetation before it was changed to coal.

From the Federal Creek field, southwest to the Gallia field, the Pittsburgh coal is of less importance but is mined by farmers for local use. Thus, in the southwest corner of Canaan township, Athens County, are several ridges and hills in which the coal has a maximum thickness of 6 ft., with the structure characteristic of that in the Federal Creek field.

Meigs County lies south of Athens and the character of the Pittsburgh coal is similar to that in Athens County. In two townships only, Scipio and Bedford, is the coal even of local value. Bedford township has a large acreage of this fuel, where it is commonly known as the "4-ft. bed," though 3-ft. would be more appropriate. Scipio township, which lies east of Bedford, has the Pittsburgh coal with a thickness of 2 to 4 ft. in its northern half, but it carries much pyrite and is not much used by the farmers. Farther south in this county the Pittsburgh coal becomes thin and is of little or no value, and the Pomeroy bed comes into prominence. The following record taken in section 20, Bedford township, shows the relative position of these two beds:

	Feet	Inches
Sandstone		
Shales	2	0
Coal, Pomeroy, No. 8a	1	6
Shales, blue	8	0
Shales with nodular limestone	11	0
Shaly sandstone	9	0
Coal, Pittsburgh, No. 8	2	6

The Gallia field is the third and least important field of the Pittsburgh coal in Ohio. It is restricted to five townships in Gallia County. Formerly coal from this field was shipped, by river, to Cincinnati and other places, but this was discontinued years ago. The principal mining centers have been the valley of Ohio River and the valley of Swan Creek, Big and Little Bull Skin Creeks, and that of Yellow Creek. The coal which is very patchy is locally known as the Swan Creek, Lewis, or Jeffers bed.

Composition of Pittsburgh Coal

The Pittsburgh coal is all of high class or rank and of high grade. The analyses given bring out clearly the reduction in fixed carbon and gain in volatile matter in going from east to west.

Pennsylvania.—The analyses in Table 1, made by the U. S. Bureau of Mines, are selected from many hundred as fairly typical of the coal of the several basins.

Table 1.—*Analyses of Pittsburgh Coal in Pennsylvania*

	Laboratory No.	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	B. t. u.
Northern Pittsburgh	2,080	3.7	34.0	56.8	5.5	1.4	13,870
Northern Washington County	17,082	2.8	35.2	55.3	6.7	1.1	13,620
Southern Washington County	1,967	2.9	33.7	58.0	5.4	1.1	
Eastern Greene County	83,777	3.0	35.2	55.9	5.9	1.0	13,650
Western Westmoreland County	4,351	2.8	32.2	58.7	6.3	1.0	
Western Fayette County	1,968	4.1	32.4	54.0	9.5	1.6	13,270
Greensburg basin	1,942	2.7	30.3	57.8	9.1	1.3	13,610
Connellsburg basin	4,411	2.4	29.9	60.5	7.2	1.0	
Latrobe basin	85,344	2.5	26.8	59.5	11.2	1.4	13,270
Ligonier basin	1,994	3.3	23.0	62.5	11.2	1.8	13,380
Southern Somerset County	81,001	3.4	21.1	65.1	10.3	1.8	13,400
Southern Somerset County	71,063	2.7	20.9	70.3	6.1	0.8	14,160

The coal of Allegheny County and northwestern Washington County is commonly classed as steam coal; that of central Westmoreland County and central Washington County as gas coal; that of eastern Greene, western and central Fayette and eastern Westmoreland Counties as coking coal. The coal of the Pittsburgh bed of Somerset County is classed as semi-smokeless or "Georges Creek" coal.

Maryland.—The analyses in Table 2, made by the U. S. Bureau of Mines, have been selected as typifying the quality and character of the coal in the several parts of the Georges Creek field:

TABLE 2.—*Analyses of Pittsburgh Coal in Maryland*

Locality, Bed, etc.	Sample			Proximate				Ultimate					Air Drying Loss, Per Cent.	Caloric Value	
	Laboratory Number	Kind	Condition	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash Per Cent.	Sulfur, Per Cent.	Hydrogen, Per Cent.	Carbon, Per Cent.	Nitrogen, Per Cent.	Oxygen, Per Cent.		Calories	British Thermal Units
Allegany Co.															
Eckhart, Ocean, No. 3½ mine..........	8,843	..	1	2.7	14.5	74.0	8.80	1.00	4.44	79.21	1.69	4.86	1.9	7,725	13,910
Frostburg, Ocean, No. 3.............	8,840	A	1	3.2	14.5	75.6	6.7	0.92	4.51	80.99	1.77	5.11	2.2	7,835	14,100
Lord, ½ mile west of; Ocean, No. 7 mine	4,386	C	1	3.42	17.65	71.84	7.09	0.84	4.72	80.17	1.45	5.73	2.7	7,868	14,162
Midland, Ocean, No. 8 mine............	8,835	..	1	2.8	14.0	75.5	7.7	0.99	4.51	80.62	1.81	4.39	1.8	7,830	14,100

West Virginia.—Table 3 collated from publications of the West Virginia Geological Survey, by Mr. Tucker, gives the average proximate and ultimate analyses of the Pittsburgh coal from the several counties of the state.

TABLE 3.—*Average Analyses of Pittsburgh Coal in West Virginia*

County	Proximate				Common to Both		Ultimate				Calorimeter B. t. u. for 1 Lb. of Coal	Calculated B. t. u. for 1 Lb. of Coal	Fuel Ratio (Carbon Divided by Oxygen plus Ash)
	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Phosphorus, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	Carbon, Per Cent.	Hydrogen, Per Cent.	Oxygen, Per Cent.	Nitrogen, Per Cent.			
Barbour	1.16	37.31	55.23	0.007	6.30	2.58	.....	....	.....	....	......	......	....
Braxton	2.09	39.68	51.78	0.009	6.45	2.11	74.27	5.14	9.71	1.10	13,451	13,337	4.37
Brooke	1.99	34.73	55.39	.....	7.63[a] 7.89[b]	1.75[a] 2.97[b]	73.61	6.10	10.83	1.08	13,298	13,174	3.99
Cabell	2.39	42.22	48.60	0.027	7.27[a] 6.79[b]	3.40[a] 3.04[b]	70.40	5.40	12.69	1.03	12,750	12,620	3.03
Calhoun	1.19	34.89	56.95	0.0095	6.97	3.41	74.19	5.61	8.75	1.07	13,744	13,729	4.72
Gilmer	2.45	41.46	49.02	0.028	7.07	2.28	75.10	5.40	9.12	1.03	13,705	13,660	4.64
Hancock	2.23	34.50	55.33	0.007	7.94	3.80	74.05	5.05	9.52	1.06	13,179	13,174	
Harrison	1.63	38.24	53.53	0.023	6.60	2.94	76.04	5.06	8.14	1.22	13,800	13,695	5.16
Kanawha	2.52	40.07	51.34	0.022	5.52[a] 6.19[b]	1.62[a] 1.93[b]	74.58	5.21	11.76	1.31	13,559	13,230	4.15
Lewis	1.09	41.59	51.08	0.0262	6.24	2.86	.....	....	.....	....	......	......	....
Marion	1.34	36.63	55.62	0.023	6.41	1.91	76.17	5.21	8.93	1.37	......	......	....
Marshall	2.13	37.40	50.97	0.008	8.49	4.26	72.00	5.91	8.27	1.06	13,353	13,666	4.32
Mason	5.43	38.79	49.33	0.029	6.45	1.79	71.07	5.17	14.35	1.17	12,883	12,506	3.42
Mineral	2.11	19.82	71.86	0.0246	6.21	0.92	79.89	4.43	6.87	1.68	13,869	13,904	6.11
Monongalia	1.34	36.63	55.62	0.023	6.41	1.91	76.17	5.21	8.93	1.37	14,048	13,705	4.97
Ohio	1.81	36.38	54.52	.....	6.90[a] 7.29[b]	1.98[a] 3.35[b]	72.56	5.86	11.66	1.04	13,385	......	3.91
Preston	0.48	30.44	63.60	0.031	5.48	1.10	.....	....	.....	....	......	......	....
Putnam	3.70	38.53	51.40	0.031	6.37	2.08	72.85	4.97	12.48	1.25	13,292	12,796	3.87
Roane	1.13	37.02	54.93	0.044	6.91	2.01	75.07	5.24	9.54	1.23	13,551	13,510	4.56
Taylor	1.34	36.63	55.62	0.023	6.41	1.91	76.17	5.21	8.93	1.37	......	......	....
Upshur	0.85	41.37	51.23	0.007	6.55	3.08	.....	....	.....	....	......	......	....
Wayne	2.66	38.30	48.35	0.0115	10.69+	3.57	67.48	4.49	12.84	1.13	12,135	11,760	2.89

[a] Ultimate. [b] Proximate.

Ohio.—The composition of the Pittsburgh coal in the several fields of Ohio is shown in Table 4 from analyses by Lord and Somermeier. Each analysis shows the number of samples used, the average (1), maxi-

TABLE 4.—*Analyses of Pittsburgh Coal in Ohio*

	No. of Sample	Aver. or Not	Moisture, Per Cent.	Vol. Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	B. t. u.
Belmont field....	24	(1)	4.13	36.15	50.78	8.81	3.77	12,818
		(2)	6.54	38.65	54.08	11.01	5.09	13,212
		(3)	2.79	32.40	46.99	5.97	1.35	12,355
Federal Creek field.	4	(1)	5.94	37.82	46.77	9.47	4.18	12,054
		(2)	6.87	40.55	48.79	11.49	4.88	12,299
		(3)	4.51	35.05	44.39	8.00	3.41	11,893
Gallia field......	4	(1)	6.84	35.35	47.35	10.47	4.45	11,670
		(2)	7.83	36.76	48.26	13.03	5.21	11,849
		(3)	5.80	34.15	45.90	9.03	3.89	11,441

mum (2), and minimum (3) of each item of the analysis. The average analyses show that the coal of the Belmont field is the best grade and that the coal in the Gallia field is the lowest. The best coal is in the eastern part of the Belmont field, as shown by the following data:

PART OF FIELD	MOISTURE, PER CENT.	ASH, PER CENT.	SULFUR, PER CENT.	B. T. U.
Eastern, average of 10 samples............	3.6	8.1	3.74	13,000
Middle, average of 6 samples..............	4.8	9.0	3.35	12,650
Western, average of 5 samples.............	4.8	9.9	4.02	12,500

HISTORY AND USES OF PITTSBURGH COAL

Because of its prominence in outcrop and its excellent character and quality, the Pittsburgh bed has been mined from very early days. Mining in Pennsylvania dates back to 1760 or earlier. By 1800, coal mining around the city of Pittsburgh had assumed some importance and coal from this bed was being used for steam making, salt evaporation, glass making, and other minor frontier industries. By 1814, it was estimated that a million bushels were being consumed in Pittsburgh; the shipment of coal from Pittsburgh began in 1803. The first locks on the Monongahela River were constructed in 1841 and completed to Brownsville in 1844. From these small beginnings has grown the vast coal mining industries on the Pittsburgh bed of today.

In Maryland, the Pittsburgh bed was discovered near Frostburg in 1782 and shipments began in 1820. The construction of the Baltimore & Ohio Railroad, in 1842, and of the Chesapeake & Ohio canal, in 1850, led

to the rapid development and increased use of the Pittsburgh coal from the Georges Creek field.

In the Panhandle region of West Virginia, the Pittsburgh bed was known at the beginning of the 19th century. By 1810, coal was being used in Wheeling for blacksmithing; mining along the Ohio gradually increased from that time. In 1836, William B. Rogers for the state of Virginia studied and reported on the "main coal of northern Virginia in the Monongahela Valley." Lack of transportation long delayed the large scale exploitation of this bed except along the Ohio River. By 1873, the first year for which definite statistics are available, the total production of coal in West Virginia was only 672,000 short tons. Recalling how much of this may have come from other beds and other sections of the state, it is clear that the present great production of the coal from the Pittsburgh bed of West Virginia is a matter of relatively recent history. For example, Monongalia County, which in 1923 produced over 17,000,000 tons, in 1900 produced only 87,000 tons. Marion County, in the same year, produced 750,000 tons, as against 17,250,000 tons in 1923.

In Ohio, the Pittsburgh coal in the Belmont field must have been known to the earliest settlers. Probably the fuel was used for domestic purposes as early as 1825; it is said to have been shipped on the Ohio River as early as 1835, and this industry became important by 1845, the coal going as far as New Orleans. Bellaire at an early date was a regular coaling station for river boats. The first railroad mine is said to have been opened, in 1858, near Bellaire. The Cleveland & Pittsburgh, Cleveland, Lorain & Wheeling, and Wheeling & Lake Erie Railroads gave an outlet to Lake Erie and mining developed at a rapid rate. In 1905, Belmont County became the largest producer in Ohio, and its lead has since greatly increased. The production, in 1924, was as follows:

Coal Mined in Belmont Field, Ohio, in 1924

	Tons
Belmont County	10,934,110
Jefferson County	4,402,982
Harrison County	2,534,680
Total	17,871,772

As in that year the production for Ohio was 30,473,007 tons, more than 50 per cent. came from the Belmont field, and nearly all of this from the Pittsburgh bed. With normal working conditions, the lead of the Belmont field should increase rapidly during the coming decade.

Coal from the Pittsburgh bed outside of Pennsylvania has been used almost exclusively for steam making and household heating. In West Virginia, the coal of this bed contains too much sulfur for first-class coke. There are some considerable areas where the resulting coke will contain but a slight fraction above 1 per cent. of sulfur, and generally 2 to 4 ft.

of the bed, when it has its normal thickness of 7½ to 8 ft., may run low enough in sulfur for a good commercial coke, but it is not practical, as a mining proposition, to separate the low sulfur portion of the bed from that which runs higher, especially in view of the present low prices for bituminous coal.

Frank Haas reports that approximately 40 per cent. of the Consolidation Coal Company's production of Pittsburgh coal from the Fairmont region of West Virginia goes as railway locomotive fuel; the other 60 per cent. is used for stationary steam boilers, gas coal, producer gas, coking, cement manufacture, and general fuel purposes.

In Ohio, Pittsburgh coal has always been used for general heating and for steam generation. For these purposes, it takes high rank though it is a smoky coal and coking tends to cut off the draft. It is also a high-grade gas coal though it has not been extensively used in Ohio for this purpose within the last 40 years, owing to an abundant supply of natural gas. The coal has been used for coke. A dozen ovens were formerly in operation a few miles west of Bridgeport, Belmont County; a more important location was Utley, Athens County, where 125 ovens were built and operated, but they were abandoned about 30 years ago. Here, for financial reasons, slack coal was used. A few miles north, at Lathrop there were 50 ovens, but these have fallen into decay. It is claimed that the coke found a ready market but that the shipping facilities were inadequate; however, for metallurgical purposes the Pittsburgh coal of Ohio is too high in sulfur to yield a first-class coke.

The Pittsburgh bed in Pennsylvania has served five principal uses—steam making including locomotive use, coke making, in metallurgy, gas making, and household use. Among minor uses has been cement making, lime burning, brick and tile burning, glass making, and a great number of minor industrial uses.

The manufacture of coke from the Pittsburgh bed at Connellsville began in 1841; it was first used in a Pittsburgh blast furnace in 1860. The extensive use of Pittsburgh coal for coking, however, dates from about 1865. There had existed a small trade before that, serving the Pittsburgh market, but from 1865 to 1875 the coke industry had an enormous growth. In 1869, coke first led charcoal in making pig iron. So suitable has the coal from the Uniontown-Connellsville basin been for making coke that for many years the coal of that basin has gone almost exclusively into the coking trade. Later, it was discovered that the coal of western Fayette and eastern Greene Counties was also suitable for coke making, so that the Monongahela has become a great artery of coal for coke making.

Farther down the Youghiogheny River, the coal was early found to be highly suited for gas making; and for scores of years "Yough coal" was shipped all over the country for gas making.

The bulk of Pittsburgh coal is admirably suited not alone for making by-product coke but for successful use in low-temperature distillation. Experimental work in this line on a commercial scale has been in progress in West Virginia for some time. The successful conclusion of these experiments will see the beginning of the end of consuming good bituminous coal for fuel in its raw state; will see the utilization of all of its valuable by-products. When this is accomplished, the vast tonnage of this great coal bed remaining unmined in four states will probably be treated by the new process near the mine mouth and only the finished products, fuel coke, oil, gas, and other valuable by-products will be transported to other regions and states.

Production and Exhaustion of Pittsburgh Coal Bed

Accurate statistics in which the production from the Pittsburgh bed is separated from the production of other beds in the same region do not exist. By assuming that all the coal from certain counties has come from the Pittsburgh bed, or making estimated allowance for coal from other beds, it is possible to estimate closely the total production in Pennsylvania, West Virginia, and Ohio. In Maryland, at present, about one-third of the mines are mining in the Pittsburgh bed; lacking closer figures, it may be assumed that about one-third of the present production of the state is from that bed. In 1910, accurate figures show that about three-fourths of the prodution of Maryland came from the Pittsburgh bed. This ratio has, however, been rapidly changing due to the approaching exhaustion of the coal in that bed on first mining. In general, it may be estimated that the total production to date from this bed has been about as follows:

Production of Pittsburgh Coal, to Date

	Tons
Pennsylvania	2,500,000,000
West Virginia	400,000,000
Maryland	175,000,000
Ohio	400,000,000
Total	3,475,000,000

Assuming an average past recovery of 60 per cent. indicates a total exhaustion of about 5,800,000,000 tons.

Reserves of Pittsburgh Coal

Reese[3] recently calculated the reserves of the Pittsburgh and other beds of southwestern Pennsylvania. His results for the Pittsburgh bed are as follows:

[3] John F. Reese: Bituminous Coal Reserves in Pennsylvania, Pa. Top. & Geol. Surv., *Bull.* No. 54 (1922), 6.

RESERVES OF PITTSBURGH COAL BEDS IN SOUTHWESTERN PENNSYLVANIA
(Recoverable coal only)

	Tons
Allegheny County	280,100,000
Armstrong County	2,800,000
Beaver County	4,000,000
Fayette County	919,300,000
Greene County	2,831,453,650
Indiana County	21,200,000
Somerset County	16,100,000
Washington County	3,516,860,000
Westmoreland County	538,300,000
Total	8,130,113,650

TABLE 5.—*Area and Quantity of Pittsburgh Coal in West Virginia*

County	Acres	Average Thickness of Coal (Assumed) Ft.	Short Tons
Barbour	22,261	6 to 10	297,236,016
Braxton	52,940.8	3 to 3½	311,369,074
Brooke	39,000	4½ to 5	330,111,760
Cabell	8,860	2–10–⅛″	44,167,156
Calhoun	11,852.8	2	41,304,637
Clay	6,809.6	2½ to 3	35,227,146
Doddridge	58,496	6	611,540,582
Gilmer	69,120	3 to 5	452,187,648
Grant	32	11	613,325
Hancock	670	5	5,837,040
Harrison	159,488	6	1,667,351,348
Kanawha	29,492	3 to 4	252,418,003
Lewis	106,137.6	1 to 6	865,501,655
Marion	149,325	7	1,824,089,959
Marshall	200,960	6	2,100,916,224
Mason	42,880	4½	336,213,504
Mineral	1,971.2	12	41,215,426
Monongalia	123,801	7	1,507,144,037
Ohio	64,000	5	557,568,000
Preston	1,059.2	10	*18,455,500
Putnam	44,800	5½	429,327,360
Roane	47,411.2	2	165,218,549
Taylor	9,024	7	110,063,908
Tucker	5	11	*95,832
Upshur	9,088	4	63,339,725
Wayne	5,120	2⅔	23,909,858
Wetzel	152,320	6	1,592,414,208
Average	....	5.543	
Total	1,416,924.4		13,684,837,480

* Mined out.

Table 5, prepared by R. C. Tucker, gives the area and quantity of Pittsburgh coal in West Virginia.

The total area in round numbers (1,417,000 acres) and tonnage (13,685,000,000 short tons) take no account of approximately 400,000,000 tons already produced from West Virginia mines. Because of lack of statistics on the Pittsburgh coal production prior to recent years, it is impossible to estimate exactly the amount that has been mined, but the amount can hardly exceed 400,000,000 short tons and may be considerably less. Assuming a 60 per cent. recovery, 666,000,000 tons of the original deposit is now exhausted. If 75 per cent. of the balance (13,018,-000,000 tons) can be mined, West Virginia's recoverable coal in the Pittsburgh bed is about 9,750,000,000 tons.

In 1917, Clark[4] published computations of the coal originally in the ground for the several beds of Ohio; his results for the Pittsburgh bed is given in Table 6. Subtracting 640,000,000 tons, as representing

TABLE 6.—*Original Deposit of Pittsburgh Coal in Ohio*

County	Average Thickness		Short Tons
	Feet	Inches	
Athens	4	2	552,000,000
Belmont	4	11	2,690,000,000
Gallia	3	5	158,000,000
Guernsey	3	8	84,000,000
Harrison	4	11	623,000,000
Jefferson	4	11	623,000,000
Lawrence	1	3	29,000,000
Meigs	3	4	173,000,000
Monroe	4	0	1,475,000,000
Morgan	4	4	100,000,000
Muskingum	2	6	29,000,000
Noble	3	9	22,000,000
Washington	2	5	334,000,000
Total			6,892,000,000

the total recovery and wastage, and estimating future recovery at 75 per cent., would give as a possible recoverable reserve of the Pittsburgh bed in Ohio about 4,666,000,000 short tons, though any such computations must not be taken too seriously.

The authors are not in a position to estimate the reserves of the Pittsburgh bed in Maryland. From a general knowledge of the character of mining in the past, a rough guess suggests that under the stimulus of

[4] J. A. Bownocker and F. R. Clarke: Coal Fields of Ohio with a computation of the original contents of the fields. U. S. Geol. Sur. *Prof. Paper* 100(*b*), (1917) 88–96.

much higher prices, that is higher in proportion to general commodity values, 100,000,000 tons may yet be mined.

RECOVERABLE COAL IN THE PITTSBURGH BED

	SHORT TONS
Pennsylvania	8,130,000,000
West Virginia	9,750,000,000
Ohio	4,666,000,000
Maryland	100,000,000
	22,646,000,000

At the recent rate of recovery, this bed should last about 180 years.

DISCUSSION

F. HAAS, Fairmont, W. Va.—Is there any reason for classifying the Berlin coal in Somerset County as Pittsburgh coal?

G. H. ASHLEY.—Yes; we originally thought that the Conemaugh formation was of fairly uniform thickness over its whole outcrop in the state. It is about 600 ft. thick in the Pittsburgh region, and is supposed to run about the same in the Ligonier region. But eastward there is a sudden thickening of 300 ft., that is from 600 to 900 ft.

There is an outcrop of Pittsburgh coal, of only a few acres, in northern Somerset County. A drill hole a this point has been put down through the Pittsburgh coal to the Upper Kittanning coal. The interval, as I remember, was 890 ft.—very close to 900 ft. to the Upper Freeport, which is at the base of the Conemaugh, or 1000 ft. to the Upper Kittanning. This thickness is well substantiated in western Maryland. A study by Swartz indicates that the lower part of the Conemaugh does not change with this extra thickness, which comes in a group of three members —the sixth, seventh and eighth members, in Maryland and Somerset Counties, which thin out before reaching the Ligonier region.

J. D. SISLER, Harrisburg, Pa.—The Conemaugh is practically 900 ft. thick in Somerset County. The Second Pennsylvania Survey thought it was approximately 600 ft. thick, and this addition of 290 ft., to be exact, has led us to believe that the Pittsburgh bed continues into the Pine Hill Basin from its southerly location west of Meyersdale.

I traced this bed through the entire basin, using the limestone beneath, and the outcrop is practically continuous; so that, to my mind there is no doubt they are one and the same bed. At Meyersdale, a small upthrow in the structure has caused some geologists to believe that it missed the hilltop, but instead it dips into the hilltop; a reversal of the structure on the anticline is enough to cause a downthrow into the Pine Hill Basin farther north.

A. C. FIELDNER, Pittsburgh, Pa.—Has any attempt been made to correlate coal beds of this basin by microscopic methods of identifying

the spores. Thiessen has shown that the Pittsburgh coal bed has a characteristic spore, easily recognizable under the microscope.

J. D. Sisler.—There has been some correspondence with the Bureau of Mines concerning this and before the report is published we hope to have that as another check. Some engineers think that the Pittsburgh bed south of Meyersdale is not the same as the one at Pine Hill, because immediately west of Meyersdale the structure changes, the bed thins down, and the partings are not the same, nor do they occur in the same parts of the bed. But, such variations in deposition are common. Some Allegheny beds, such as the Freeport and Kittanning, vary within a few hundred yards so as not to be recognizable.

E. V. d'Invilliers, Philadelphia, Pa.—The thickening of the Conemaugh measures to the east was fairly well known prior to 1890. Up to that time it had been generally accepted as 600 ft. It was well known in 1890 that there was an interval of 900 ft. between the Pittsburgh and the Kittaning. I was in charge of some drilling at that time. We drilled 700 to 750 ft. in the highest hills in the center of the basin, but the Pittsburgh bed had not been discovered in the hills above. The development in the Meyersdale district is a different matter.

S. A. Taylor, Pittsburgh, Pa.—Has the lack of coal south of Belmont been correlated with the structure of the adjoining counties in West Virginia? I am inclined to think the want extends over into the No. 8 fields.

J. A. Bownocker.—I have never been able to determine why the Pittsburgh coal is generally absent in Ohio. Whether it was never deposited, or whether it was washed away before it was covered. Work has been done in that field for the past 2 years; perhaps a little later we will have some more definite information concerning it.

J. P. Williams, Pittsburgh, Pa.—I notice an area in Mercer County and one in Logan.

J. A. Bownocker.—Those are two artificial lakes.

J. P. Williams.—Is there anything to indicate the extent of the Pittsburgh bed other than has been shown?

J. A. Bownocker.—The rocks farther west lie below the Pittsburgh coal; so it is useless, in Ohio, to look for Pittsburgh coal farther west.

L. S. Panyity, Bradford, Pa.—In answer to Mr. Taylor I would say that one can find traces of the Pittsburgh coal and can locate its position by the blossoms, but it is not found in workable deposits in that area. It makes a very good "marker" or key-horizon, however, because its blossom occurs whenever it is due. A geologist working out the structure of an area will find it a very good key-bed.

Member.—Is there any way of improving, mechanically, the chemical composition of the Pittsburgh coal, so as to get the same uniform product in Ohio?

J. R. Campbell, Scottdale, Pa.—The reduction of the ash is very satisfactorily effected by washing, also of the sulfur. I have investigated nearly all processes of cleaning coal, and find the dry method can be used in certain cases of commercial coals with fair success, but, for metallurgical purposes, we must use wet washing, and generally with fine crushing, to get the sulfur reduction.

A free discharge Rheolaveur machine, able to handle 60 tons per hour of metallurgical coal, will reduce the sulfur 60 per cent in some coals.

T. B. Sturges, Pittsburgh, Pa.—My belief is that the thinning or want of the bed in the southern part of the Belmont field is connected with the thinning in West Virginia. Drilling in both Ohio and West Virginia shows that the coal thins out along this line. The coal seam thins down from its full thickness within a mile or two, sometimes less.

S. A. Taylor.—I made an investigation in the vicinity of Mason City, and found an oil well at Spencer where the drillers claimed to have found the coal, but could not get any definite data on the coal.

Down the Ohio river, 3 or 4 miles below Gallipolis, there is a thin (3½-ft.) outcrop of Pittsburgh coal in a little stream. The bed is erratic in the hill sections. Tracing the strata back up the river, strengthens the belief that the Pomeroy coal is the Redstone seam.

Downward and across the Kanawha river (from Plymouth) the coal is broken up in this large field and beyond it is very sporadic. The same thing is true in Cabal County, where the Pittsburgh coal is not workable, being 2 to 3½ ft. thick.

As we meet the anticlinals, from the great breaks in the mountains, the coals decrease in volatiles. The transition is gradual until the high-grade gas coals in the Irwin field are reached. A portion of the Pittsburgh bed is below sea-level in Greene County, Pa., the only instance of this occurrence in this country so far as I know.

If you draw a line through Pocahontas, the mouth of Elk River, and a little east of Pittsburgh, you will include all the veins of coal in the Appalachian field. It is the only point in the whole Appalachian chain, that I know of, where one cross-section will cut all the seams in the Appalachian Mountains.

L. S. Panyity.—It was our task to determine what the geological structure at Pomeroy was, we were interested in the general structure of the country and placed most of our observations on the dip of the strata. We were confined in our work to the Ames limestone and beds above the Ames. The observations were based upon the difference in elevation and

the intervals above the Ames limestone, and these observations would indicate that Dr. Bownocker's views have some merit.

A certain amount of Pittsburgh coal crops out farther west. In following it through by means of its dip, it was determined to underlie the Pomeroy coal. The fact that, in Pennsylvania, a heavy sandstone may overlie the coal similar to the sandstone cover of the Pomeroy coal, is not a definite correlation, because in many places where mined or stripped at the surface the Pittsburgh coal does not show any sandstone cover that would resemble the Pomeroy sandstone.

I agree that the Pomeroy coal may be correlated with the Redstone coal, basing my opinion on the geological section of the area, especially on intervals above the Ames and Cambridge limestones.

Time Element in the Control of Face Conditions in Coal Mining

By H. F. McCullough,* Scottdale, Pa.

(Pittsburgh Meeting, October, 1926)

The success of a coal-mining venture as relates to operations at the gob or break-line, such as the drawing of pillars or the working of long-faces, depends upon the control of face conditions. The measures required to control the latter are dependent upon the action of the coal seam, roof and floor at and in the vicinity of the faces, but the manner in which these will act is dependent upon the rate of face movement as well as upon the face characteristics and upon the forces exerted. Yet the merits of a method of working—the practicability of controlling face conditions and the means required for such control when applied to any particular case—are usually judged only upon a consideration of the characteristics of coal seam, roof and floor, and the manner in which they have been observed to act.

Practically every coal-mining man knows that face conditions are influenced by the rate of face movement, but few have appreciated that this is as much of a factor in the control of face conditions as are the face characteristics and the forces exerted. Fewer yet have recognized or taken advantage of faster rates of face movement as one of the principal means of establishing control of face conditions or of securing more favorable face conditions. Seemingly inherent difficulties, particularly in the control of roof conditions, but also in the control of conditions of the coal seam and floor, are frequently mitigated or disappear when the rate of movement of the faces is sufficiently increased.

The characteristics of the coal seam, roof and floor and the forces exerted upon them do determine the general trend or tendency of face action, but the manner in which the coal seam, roof and floor will act and be acted upon and the magnitude of the action during the time the site of mining operations remains at any particular point are very largely determined by the rate of movement of the faces.

Progressive Nature of Forces Exerted

Coal seam, roof and floor are composed of materials having definite properties or characteristics. The potential or ultimate values of the

* Engineer, H. C. Frick Coke Co.

forces exerted at the faces are determined by such factors as the depth and character of cover, the arrangement and methods of working, and the inclination of the coal seam. As these forces come to be exerted at any point they are not initiated at their maximum values; rather the magnitude of the forces exerted at any point is of a progressive nature and progresses from zero to maximum values only after an appreciable interval of time.

The facts that the extraction of the coal from any area is not immediately followed by the exertion of the full magnitude of the forces which ultimately result therefrom, and that the forces exerted do not immediately manifest themselves in the fracturing of the roof strata, the crushing of the coal seam, and the heaving of the floor, are what makes the time element an important factor in the control of face conditions. The faster the rate of movement of the faces, the less the forces set up by the extraction of the coal will have manifested themselves at any particular point before the faces have moved beyond that point. The less the forces have manifested themselves at any particular point, the more easily and effectively may the face conditions at that point be controlled. The practical advantages of the faster rate of face movement are numerous and important.

Control of Face Conditions

Roof Control

The means required for the control of face conditions are determined by the magnitude of the forces encountered at the site of mining operations. Since a faster rate of face movement keeps the site of mining operations farther removed from the region where the maximum effects of the forces exerted are manifested, it serves to facilitate control of face conditions. In general, it is only after the first major falls of roof strata into the gob have occurred that the full "weight" or pressure of the overlying strata is exerted upon the coal. Thereupon, the strata overhanging the coal face, and extending back over the coal for some distance, act as a lever and as such descend and exert a pressure upon the coal, which is manifested back over the coal for a certain distance.

Roof control becomes a problem in proportion to the degree to which the strata overlying the working space along the faces descend or deflect downward from the position occupied before the coal which underlaid and supported them was extracted. In their original undeflected position the roof strata are in the condition wherein they may be most easily controlled. The stressing and consequent fracturing of these overhanging strata increases as their downward deflection increases. But the downward deflection and the fracturing of the overhanging strata are progressive rather than instantaneous actions.

The faster the rate of face movement, the less stressed and fractured will be the roof strata overlying the working space along the faces and, hence, the more easily may roof conditions be controlled. With rapidly moving faces, the face timber can be lighter because it has served its purpose and may be moved before the overhanging roof strata has descended and fractured enough to load it heavily.

Less face timbering can be used because the site of mining operations remains at any one point a shorter length of time and, hence, the forces which gradually come to bear upon the overhanging roof strata and cause their fracturing progress less before the site of mining operations has moved on. A greater amount of timber may be recovered and the timber may be more easily recovered because during the shorter time that it is required to be left in place it is less heavily loaded than it would be with a slower rate of face movement.

There is less slate and rock to handle and dispose of because, with a fast rate of face movement, the roof strata overlying the working space along the faces are less stressed and consequently less fractured than they would be with a slower rate of face movement.

Crushing of the Coal

The coal is crushed by the descent of the overhanging roof strata. As the descent of these strata progresses with time, the sooner the site of mining operations moves on and from under any particular area of descending roof strata, the greater will be the recovery, the less will be the crushing and the greater the possibility of producing lump, the easier may the coal face be undercut, and the more effectively may it be shot.

Heaving of the Floor

With heaving floor, the magnitude of the disturbance is proportional to time and, hence, the faster the rate of face movement the less will be the disturbance, at the site of mining operations, in the working space along the faces and in the headings leading to the faces.

Concentration of Active Mine Areas

Most of the advantages obtainable from improved mining practices are dependent upon concentration of the active mine area. However, concentration of the active mine area is possible in proportion to the degree to which the rate of face movement is increased. For a given production the active mine area will be inversely proportional to the rate of face movement.

Safety

More accidents are due to falls of rock and coal than to any other cause. With rapidly-moving faces, the strata overlying the working spaces are less stressed and less fractured, hence the liability of falls is

decreased and safer working conditions created. Increasing the rate of face movement has invariably resulted in a decrease of accidents from falls.

Influence of Rate of Face Movement in Determining Control of Face Conditions

The importance of the rate of movement of the faces is frequently, or possibly generally, overlooked in the devising of mining methods and in the determination of the means required for mine working and for the control of face conditions. Opinions as to the feasibility of new methods of working or of means for use in working are most frequently based upon a consideration of the characteristics of the coal seam, roof and floor and on how they are observed to act under prevailing methods of working.

As an example: Many believe that some form of long-face working is necessary to permit realization of the full possibilities of mechanical loading equipment. It has not been found difficult to arrange the workings so as to afford opportunity for the loading equipment to work to the best advantage, but it has not been found so easy to maintain control of the face conditions which result therefrom. In fact, the problem of successful mechanical loading in long-face workings may be said to be that of maintaining control of the face conditions. Many are experimenting with long-face workings in an effort to determine the possibility of and the means required for the control of face conditions in their particular case. Hand loading is being used in most of these experiments and in most of them this and an intermittent supply of mine cars limit production and prevent the attainment of the rate of face movement which would prevail if the contemplated means of working were inaugurated.

The purpose of these experimental workings is to establish the practicability of long-face working by determining the means required for the control of face conditions; but the means required for the control of face conditions are dependent upon the rate of movement of the faces. The means found necessary to control face conditions with a slow rate of face movement, such as would almost necessarily prevail with hand loading and an inadequate means of transporting the coal away from the faces, would produce little information as to the means required to control conditions when the rate of face movement was faster.

Such experimental work is inconclusive and misleading. A fast rate of face movement is the primary object of mechanical loading. Practically all the advantages sought through recourse to mechanical loading are those which arise from the greater rate of face movement which is made possible thereby. Concentration of active mine areas, the source of most of the advantages of mechanical loading, is attained in proportion to the increase which is effected in the rate of face movement. Experi-

ence has shown that the rate of face movement greatly influences face conditions and that what would be a technical or economical failure with a slow rate of face movement may be a complete success with a faster rate of face movement.

The rate of face movement is the governing factor in the control of face conditions on faces with fixed characteristics. Hence, plans should be made and experimental work directed so as to first obtain a reasonable or near approach to the rate of face movement which is anticipated when the whole program of mechanical loading is operative. Until the anticipated rate of face movement is attained, the face conditions which would prevail with that rate of movement are not established. Until these conditions are established the means required for their control cannot be determined.

Experiment with faces moving slower than the anticipated rate results either in failure to control face conditions, or in the development or selection of means of control adapted for conditions different than those which would prevail with the anticipated rate of face movement.

Means required for the control of conditions on slowly moving faces may be more rugged and more expensive in first cost and cost of handling than would be required for the control of conditions on the same faces if they were moved more rapidly. Such means of control applied to fast moving faces not only increase the costs but may so slow down the whole procedure as to prevent rapid face movement and defeat the purpose of the whole work, viz., concentration of active mine areas. Hence, in experimental work done for determining the means required for mine working and for the control of face conditions, the first object should be to attain the anticipated rate of face movement.

Control of face conditions is not a problem in statics; it is a problem in dynamics. Matter in motion is being dealt with and time is a controlling factor in its action. In the mining of coal, the rate of face movement is not to be regarded as an inconsequential result of methods of working directed to the attainment of other ends. The rate of face movement is one of the principal means of establishing and facilitating the control of face conditions, and a governing factor in the cost of producing coal.

Experiments in Shot-firing with Low and High-voltage Currents

By A. C. Watts,* Salt Lake City, Utah

(Salt Lake City Meeting, September, 1925)

For several years, a mine in Colorado experienced considerable trouble from small fires caused by the blasting of coal. Although a well-known make of permissible powder was used, it was first thought that the powder ignited gas that might be present and that this, in turn, set fire to line brattices or to the coal. Experiments were, therefore, conducted with another make of powder, but the results convinced the superintendent of the mine, Robt. Williams, Jr., that the fires were not caused by the powder but by the high-voltage electric current used for detonating the shots.

The mine "makes" gas very freely, giving off from three-fourths to one million cubic feet of methane every 24 hours. The coal is a good grade bituminous of a very free burning quality.

Electric shot-firing from the outside, after all men and animals were outside, had been practiced for a number of years. The shooting circuit was connected with the main power circuit at the mouth of each entry. Shooting circuit switches were mounted in boxes, which were kept locked with the handle of the switch, in open position, showing through the bottom of the box. The door of the box could not be shut and locked when the shooting switch was closed.

At the entrance to each room, or working place, a small switch, called the miner's switch, was placed on the shooting line. It was the duty of the miner, when entering his place at the beginning of the shift, to see that this switch was open and that it was kept open until he left the place at the end of the shift to go out of the mine. Grounded return was used for main power lines.

The current was 500 volts, direct current, with 100 to 300 amp.

The nearest working face was 9600 ft. from the shot-firers' switch outside the mine and the most remote working face was 2½ miles. The various working faces were scattered over a wide area between these two points.

A description of the main power line would be valueless; the conductors were of sufficient size, the drop in voltage being not more than

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15 per cent. The shooting circuits were No. 8 wires in entries and No. 12 wires in rooms—double lines.

After first experiencing trouble with fires, the shot-firers were required to make an inspection immediately after detonating the shots; and to

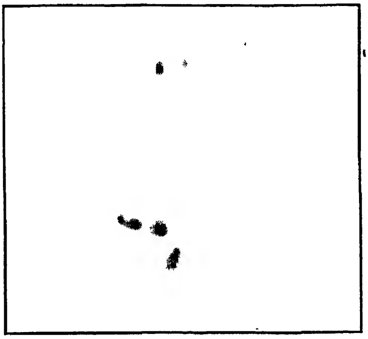


FIG. 1.—FOUR DETONATORS IN PARALLEL; CURRENT THROUGH TWO 500-OHM RESISTORS IN PARALLEL.

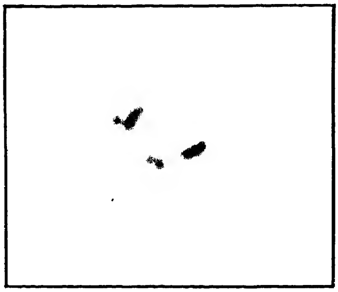


FIG. 2.—SIX DETONATORS IN PARALLEL; CURRENT THROUGH THREE 500-OHM RESISTORS IN PARALLEL.

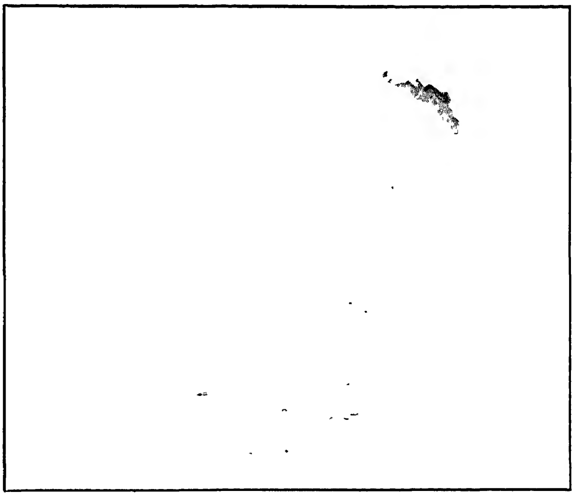


FIG. 3.—TWO STICKS "MONOBEL" POWDER; CURRENT FULL STRENGTH; HOLE 3 FT. 6 IN. DEEP; 3 FT. 4 IN. GRIP IN COAL.

expedite their work rope, trips were run. But until current was reduced, as a result of the experiments, a fire would start about every week or ten days, and at least one local explosion occurred which blew out concrete overcasts and stoppings and extended 2000 ft. from point of origin.

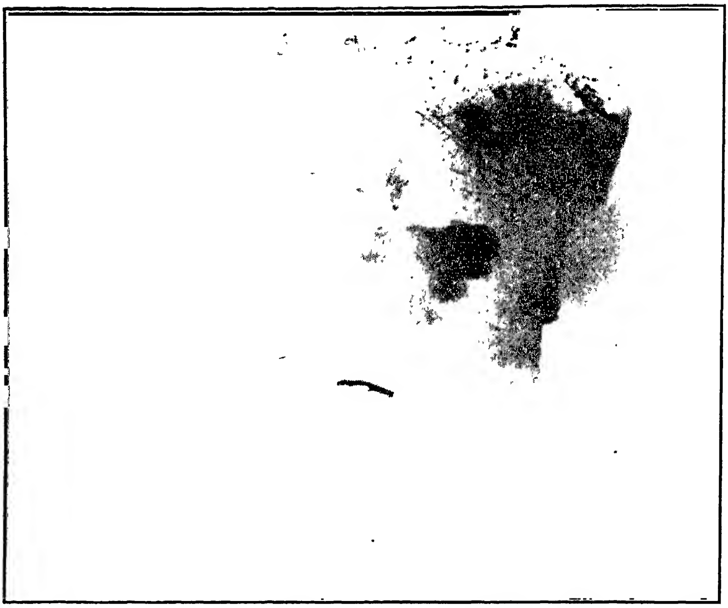


Fig. 4.—Windy, or blown-out shot; one stick "Monobel;" current full strength.

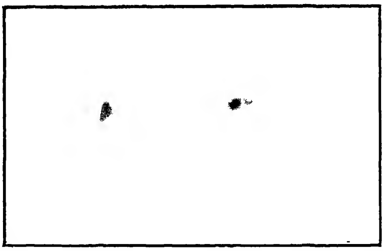


Fig. 5.—Two detonators in parallel; current through one 500-ohm resistor.

Fig. 6.—Twenty-five detonators in parallel; current through 8 in. of water.

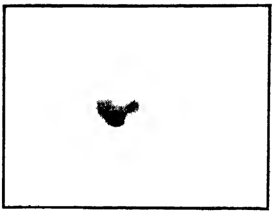


FIG. 7.—ONE DETONATOR; CURRENT THROUGH ONE 500-OHM RESISTOR.

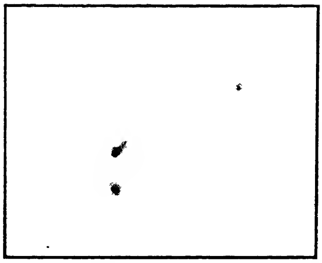


FIG. 8.—SEVEN DETONATORS IN PARALLEL; CURRENT THROUGH THREE 500-OHM RESISTORS IN PARALLEL; TWO CAPS MISSED FIRE.

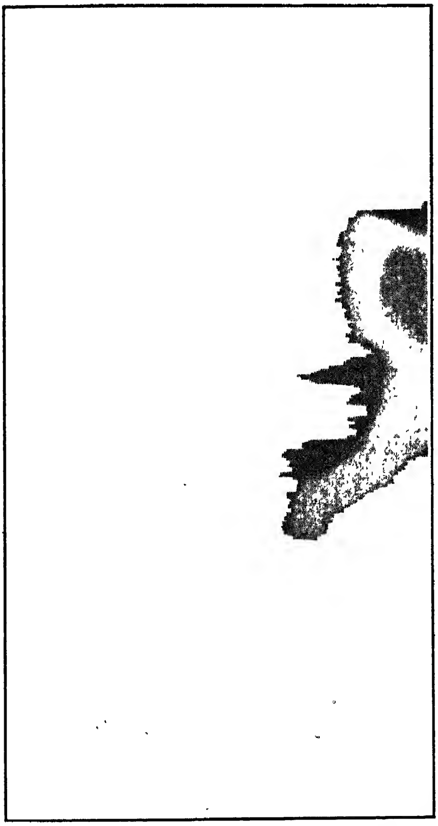


FIG. 9.—ONE DETONATOR; CURRENT FULL STRENGTH.

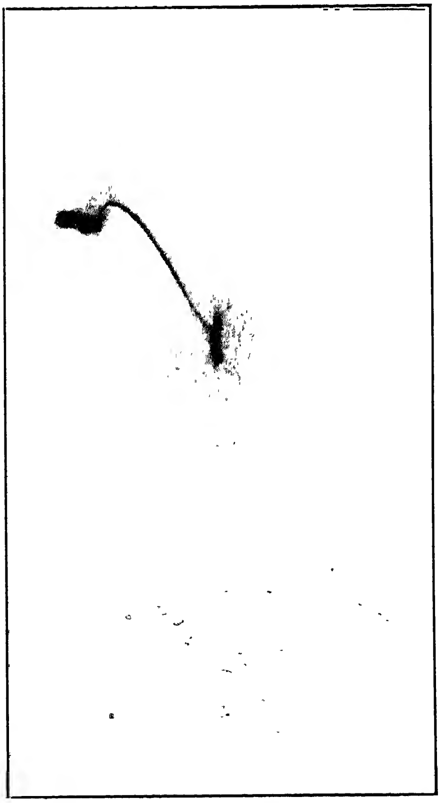


FIG. 10.—ONE DETONATOR; CURRENT FULL STRENGTH.

A more extensive explosion was prevented by the wet condition of the mine caused by efficient sprinkling.

On idle days, the superintendent and other observers experimented with other powders and shot with full current and with a hand battery

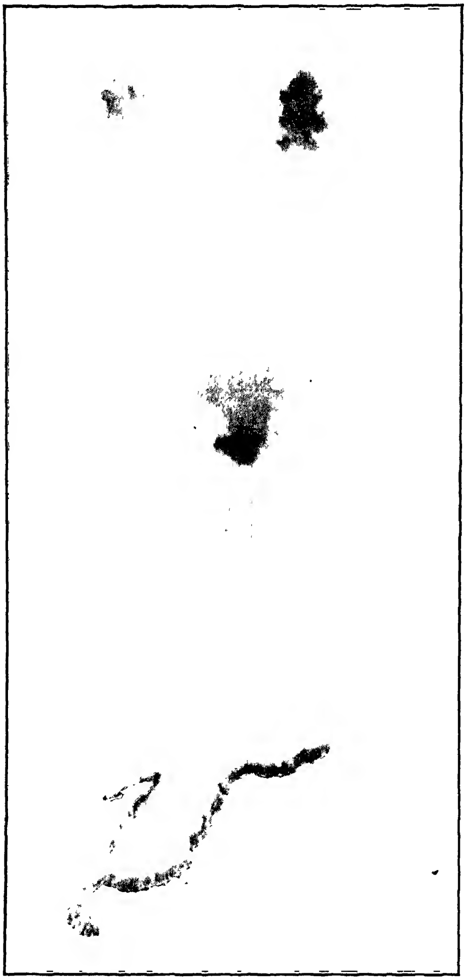


FIG. 11.—SHORT CIRCUIT IN CAP WIRE; CURRENT FULL STRENGTH.

FIG. 12.—SHORT CIRCUIT IN CAP WIRE; CURRENT FULL STRENGTH.

With the full current of 500 volts and 100 to 300 amp., flame invariably accompanied the detonation of shots. The flame of one shot was estimated to bė at least 19 ft. long. Another shot acted like a roman candle and threw sparks of fire for 20 ft., one spark lighting on some brattice cloth and burning a hole in it. When the hand battery was used no

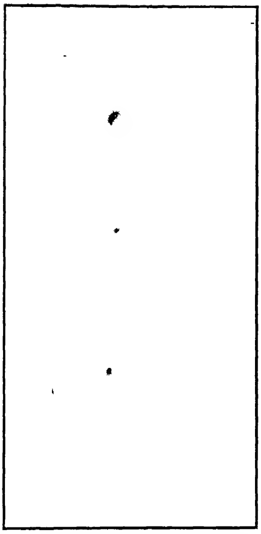


FIG. 13.—THREE DETONATORS IN SERIES; CURRENT THROUGH WATER.

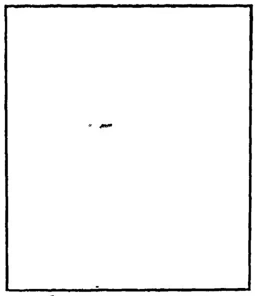


FIG. 14.—ONE DETONATOR; CURRENT THROUGH WATER.

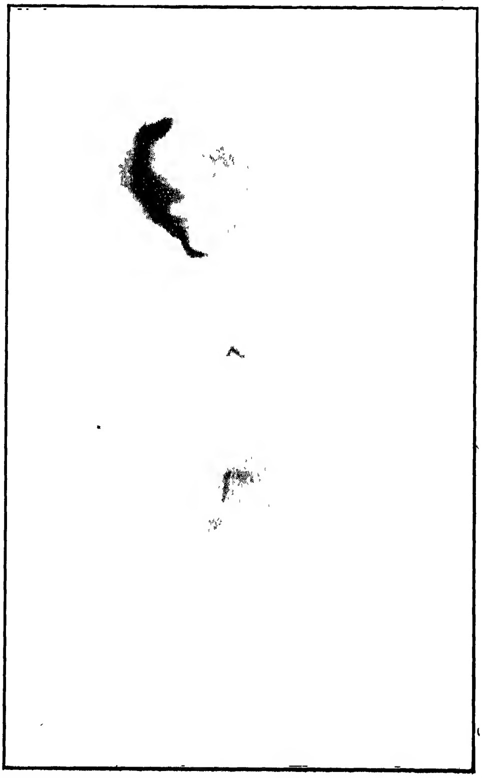


FIG. 15.—THREE DETONATORS IN SERIES; CURRENT FULL STRENGTH.

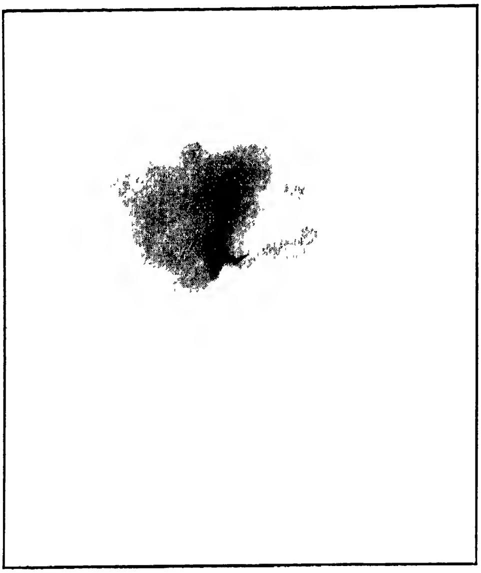


FIG. 16.—ONE DETONATOR; CURRENT FULL STRENGTH.

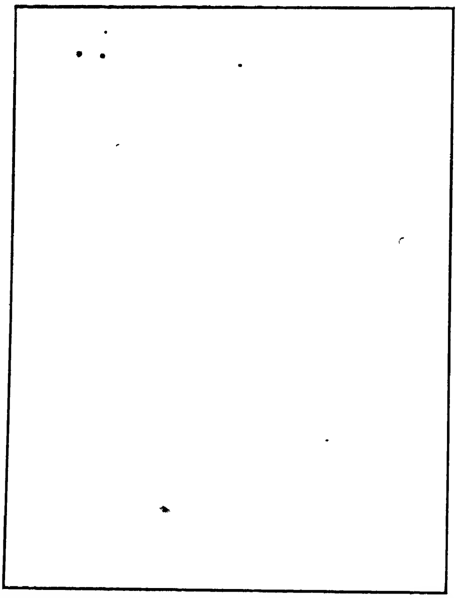


FIG. 17.—DETONATOR WRAPPED IN GASOLINE-SOAKED WASTE; CURRENT THROUGH WATER.

flame could be seen, hence it was decided to experiment further with reduced current.

A water resistance 14 ft. long, with copper terminals, was constructed. Experiments showed the proper length between terminals to use for the mine, but as this varies for each mine, the figures would be of no value. This was used until one night a shot-firer decided not to use it because he had missed several shots the previous day, which led him to believe

FIG. 18.—DETONATOR WRAPPED IN GASOLINE-SOAKED WASTE; CURRENT THROUGH WATER.

the current was not strong enough, so he used the full current with the result that a fire was started. To remove the possibility of such tampering, a resistance was made out of an old rheostat and adjusted until it was found that a current of 100 volts and 45 amp. was sufficient to detonate all shots without causing fires.

Tests were made with Form CC-500 ohms resistors. Because no recording instruments were available, no current readings could be made

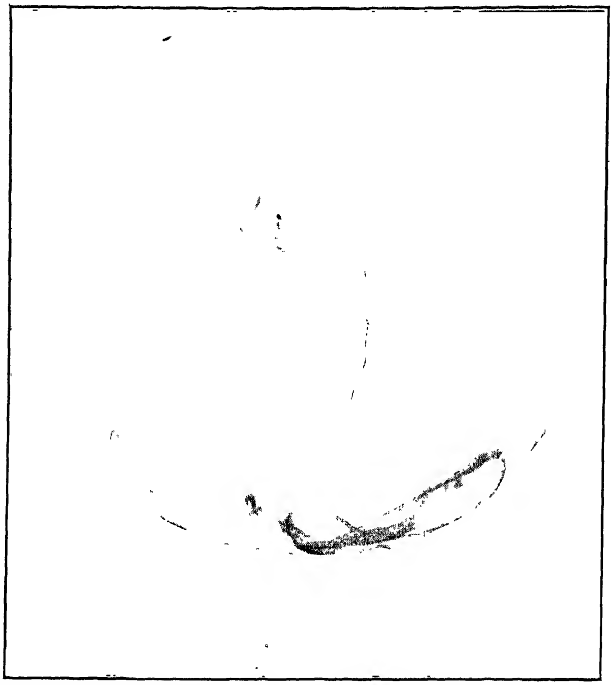


FIG. 19.—DETONATOR WRAPPED IN GASOLINE-SOAKED WASTE; CURRENT THROUGH WATER.

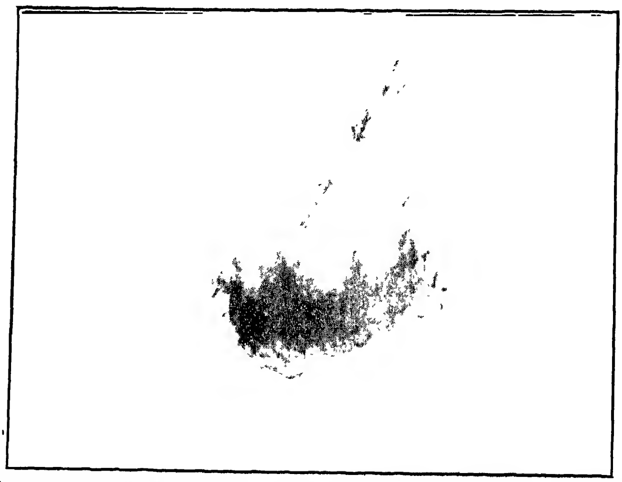


FIG. 20.—DETONATOR WRAPPED IN GASOLINE-SOAKED WASTE; CURRENT THROUGH WATER.

with the instruments used, but visual observation indicated that the flame from detonators, when resistors were used, was about the same as when the current was cut down with water resistance—a small blue flame hardly discernible and shown on a photographic plate as a pin point of light. Although it was determined that resistors could be used, they were deemed impracticable for ordinary mine work.

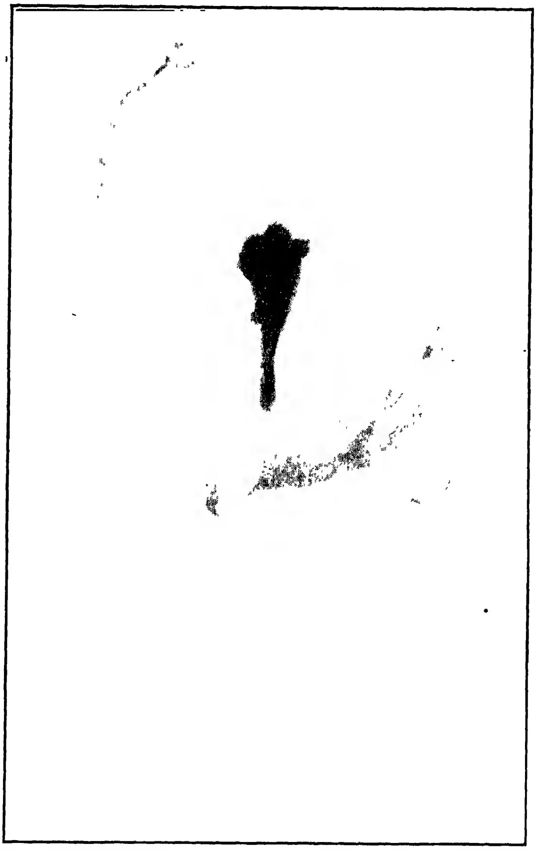


FIG. 21.—DETONATOR WRAPPED IN GASOLINE-SOAKED WASTE; CURRENT FULL STRENGTH.

When testing with detonators, it was found that with full 500-volt current the paraffine covering on the lead wires of detonators was often set on fire and sparks thrown off, which could easily ignite methane or dry brattice cloth; this did not occur when a resistor or resistance was used.

When a shot-firing system is installed, resistors can be used on the outside of each working place, or suitable resistance can be located for

each district or full resistance can be placed outside of the mine.

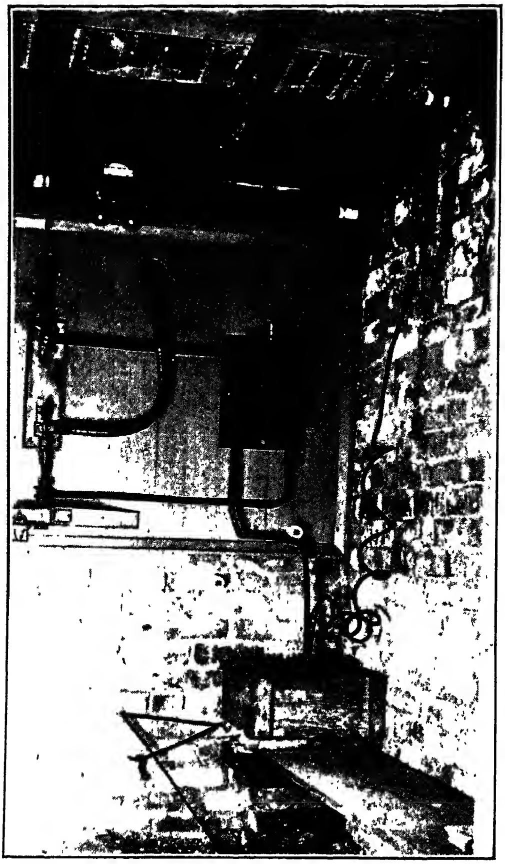


Fig. 22.

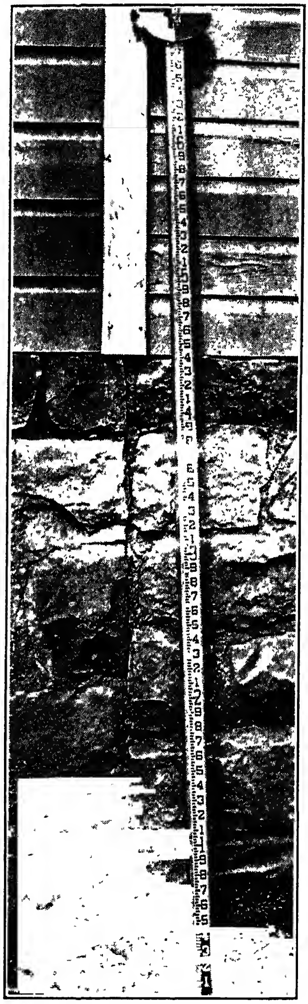


Fig. 23.

Fig. 22.—Interior of shooting cabin with switches in shooting position. Note lock on main power switch.

Fig. 23.—Level rod taken with same focus as detonator pictures, for comparing measurements of bodies of smoke and flame.

Photographs of the flame resulting from shots made with different voltages and under different conditions (Figs. 1 to 21), plainly show the effect of high voltage. Photographs showing caps detonated through resistances can scarcely be reproduced as the flame shows only as a pin point.

Mining Methods in the Pittsburgh District*

By the Pittsburgh District Sub-committee on Coal and Coke†

(Pittsburgh Meeting, October, 1926)

Early Practice

The first mention of the mining and use of coal in the Pittsburgh district refers to the mine under Duquesne Heights that furnished coal for the garrison at the fort at Pittsburgh in 1760. Coal had been dug and used as fuel just outside of Brownsville, Fayette county, in 1759, but this was not a regular mine as the opening was in an outcropping. Maps of Ohio in 1770 showed "Cole Mines" at various points. The Penns purchased all the coal south of Kittanning in southwestern Pennsylvania from the Chiefs of the Six Nations for a reported price of $10,000. During the Revolution, coal from Herron Hill, Minersville, and Coal Hill, all within the present city limits, was used in Pittsburgh. There is today one custom pit opened in Coal Hill (Duquesne Heights).

The first steam engine was set up in Pittsburgh in 1794 and used coal as fuel. The first mine in the Youghiogheny gas district was opened in 1796. In 1803, the first shipment of coal by water was made from Pittsburgh down the Ohio river and landed at Philadelphia, where it sold for approximately $9.50 per ton. The mines along the Monongahela were naturally the first to be opened. With the completion of locks and dams in 1844, permitting slack-water navigation, the industry began to thrive. The No. 8 field of Ohio and the Fairmont region of northern West Virginia were opened in the period 1840 to 1855. From that time on there has been a continuous development of the coal resources of this district, until at present productive capacity is enough to supply 30 per cent. of the country's needs of bituminous coal.

The original method of working the Pittsburgh seam was to drive under cover at some easily accessible point on the outcrop with a single entry, usually 1½ yd. wide. Rooms or gangways were turned to the right and left from this entry and ventilation was usually poor and inade-

* The Pittsburgh district includes here Allegheny, Washington, Greene, Fayette and Westmoreland counties in Pennsylvania; Belmont, Jefferson, Harrison and Monroe counties in Ohio, and the Northern and Panhandle sections of West Virginia.

† This paper was prepared by N. G. Alford and B. F. Hoffacker, from their information and from data submitted by C. F. Lynch, general superintendent, H. C. Frick Coke Co.; J. C. Lubken, general manager, Allegheny Pittsburgh Coal Co.; Frank Dunbar, general superintendent, Hillman Coal & Coke Co.; Valley Camp Coal Co.; Lincoln Coal & Coke Co. and Buckeye Coal Co.

quate. Where the coal dipped drainage was secured by rising on the coal. Much bottom and top were left in the workings and were usually lost.

Present Practice

From this simple country-bank system there has developed the present room-and-pillar method with first the double entry, and then the multiple entry system. In some parts of the district no attempt is made to draw pillars; in others, where the coal is much more valuable in place, the recovery varies as high as 95 per cent. of the mining section. Where no pillars are drawn, as in eastern Ohio, the advancing work recovers between 50 and 60 per cent. of the coal. Where pillars are drawn there is a generally uniform system in the method of advancing and retreating; and various plants, due to local conditions, frequently change this system to suit their individual requirements.

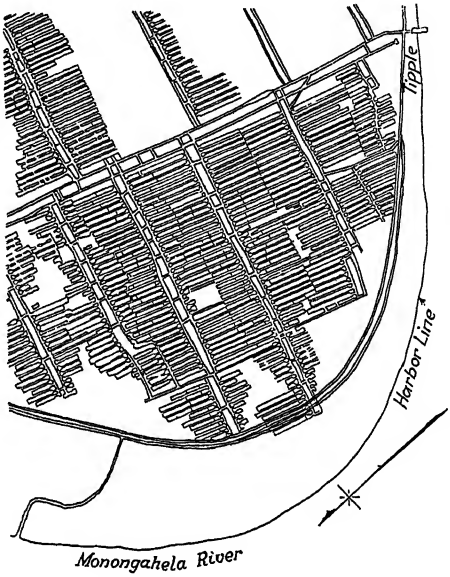


Fig. 1.—Plan of mining the Pittsburgh coal, 4th pool, Fayette county, 1860–1880.

Fig. 1 shows the method of working the Pittsburgh coal in the 4th pool on the Fayette county side of the Monongahela river that prevailed between 1860 and 1880. Fig. 2 shows the present method of advancing and retreating in the Pittsburgh seam in the Connellsville coke region. Fig. 3 shows the double-entry room and pillar method used prior to 1900 in the old Connellsville basin. Fig. 4 shows the present methods in the Ohio No. 8 district. Figs. 5 and 6 show the present method in the West Virginia Panhandle and Fairmont districts respectively, and Fig. 7, the latest practice in shaft design.

Prospecting is done chiefly with core drills, by testing at the face in adjoining workings, and by sampling carloads of coal from adjoining mines. Where possible, outcrop sections are cut, sampled and analyzed. The thickness of a section of the Pittsburgh coal seam throughout the field varies from 4½ to 9 ft. Coal in place weighs from 78 to 82 lb. per cubic foot. Mining losses are from 5 to 45 per cent. The yield per acre-foot varies from 1080 to 1600 net tons.

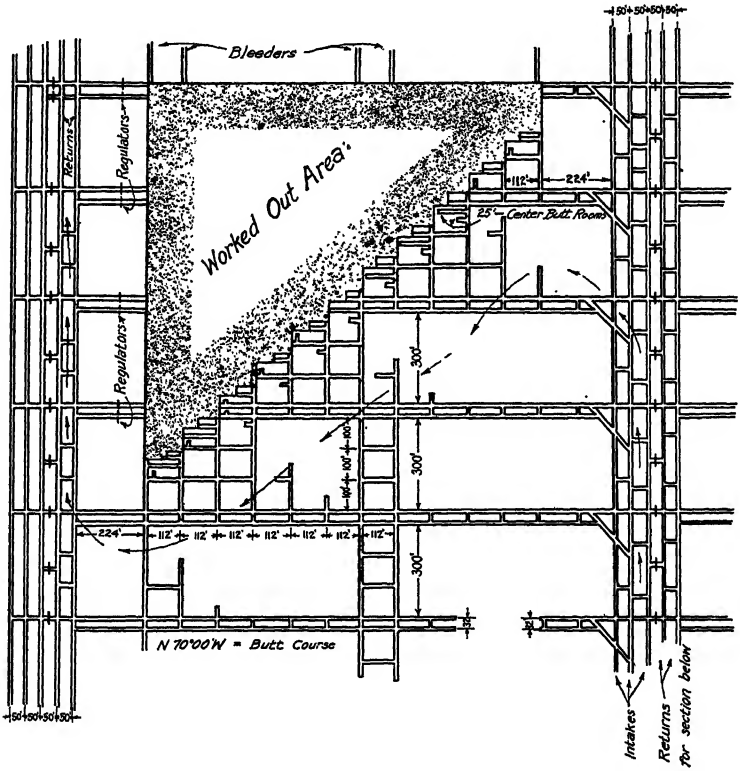


FIG. 2.—STANDARD MINING PLAN SHOWING METHOD OF ADVANCING AND RETREATING, PITTSBURGH SEAM, CONNELLSVILLE REGION.

In sections where pillars are not drawn, rooms are driven 20 to 25 ft. wide and 200 to 300 ft. long with 8 to 10 ft. of pillar between them. The rooms are often turned off butt entries and are driven with the room faces on the faces of the coal. The remainder of the field is driven mostly on the panel system with practically all work at right angles on the butt and face cleats. The standard length of rooms varies from 200 to 300 ft., the width 10 to 20 ft. Headings are 8 to 10 ft. wide, are from 2 to 6 in number, according to the ventilation and haulage requirements, and are on 35 to 50-ft. centers.

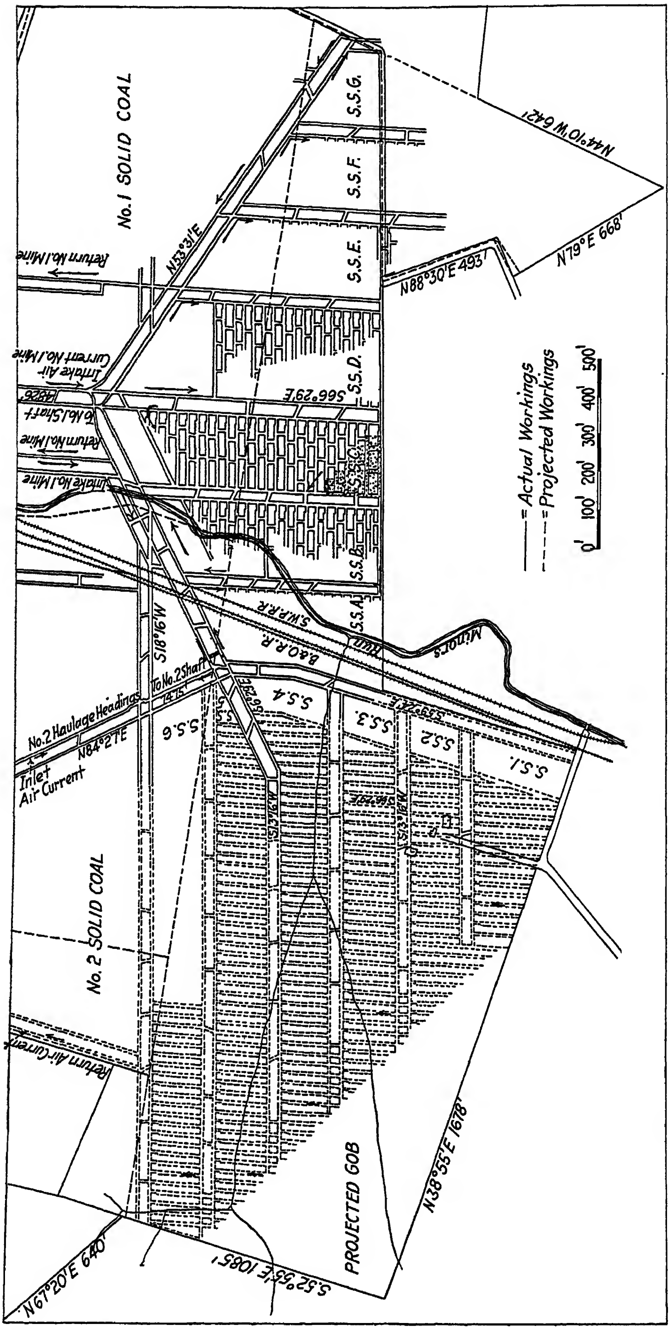


FIG. 3.—PLAN OF MINING BY THE DOUBLE-ENTRY ROOM-AND-PILLAR METHOD USED PRIOR TO 1900 IN THE OLD CONNELLSVILLE BASIN.

Holdings and Leases

The amount of coal reserves held by the various companies mining in this district differs widely. Holdings may be as small as 1 acre or as large as 152,000 acres. The preferred unit of area for one operation is from 1000 to 2000 acres, although as much as 6000 acres is worked through one opening.

Coal is acquired in irregular-shaped tracts, and the present practice is to square holdings by exchange on butt and face lines. The ownership

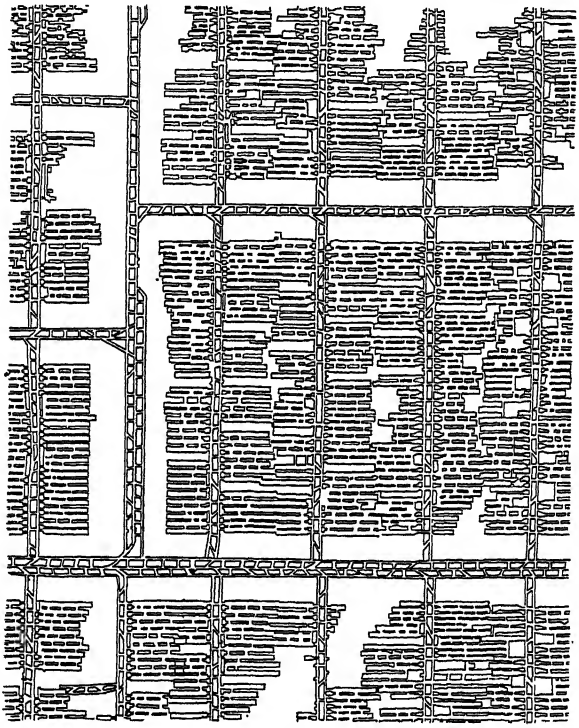


FIG. 4.—PRESENT PLAN OF MINING IN OHIO NO. 8 DISTRICT.

of the coal is practically all in fee; only a small percentage is leased. Royalties on leaseholds vary from 10 cents per net ton to what will eventually be 52 cents per net ton where the royalty is based on an annual increase during stated periods for the life of the operation.

Facilities

Practically no natural limitations, especially from a topographic standpoint, prevent development of the coal anywhere within the district. A supply of fresh water is available for all domestic and mining purposes and the district is amply served by high-tension power lines linked with some

dozen power plants. Local timber is scarce and but very little of the original or second growth still remains.

The Ohio river and its tributaries, the Monongahela and the Allegheny, afford valuable means of transportation, and four major railroad systems traverse the boundaries and the interior. In the old developed fields a great amount of pumping is required. From 1 to 15 tons of water is handled per ton of coal mined.

There are no legal restrictions or lease conditions affecting the present methods used in mining. Mines working organized labor are the only

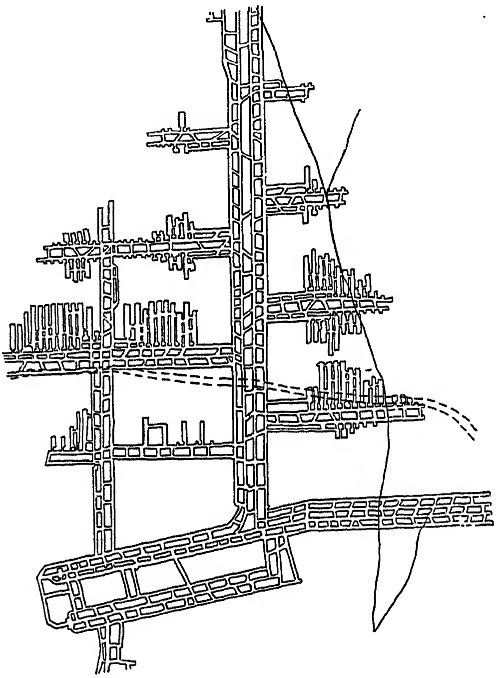


FIG. 5.—PRESENT PLAN OF MINING IN THE WEST VIRGINIA PANHANDLE.

ones having restrictions on narrow work. The district is both union and non-union. Most of the workers are of foreign birth, representing at least a dozen of the Central European countries.

CHARACTERISTICS OF THE SEAM

The Pittsburgh seam has a persistent pair of partings 3 to 8 in. apart, known as the twin slates or "bearing-in" bands, and a slate band below the twin slates that separates the brick coal from the bottom coal. The "breast" coal above the twin slates is generally the purest portion of the seam. The top coal occurs in 1 to 6 benches, separated from the main bench and each other by shale, slate or sandstone partings of vary-

ing thicknesses. In some sections one or more of the roof coals becomes thick enough to permit recovery. Faults, erosions, horsebacks, swamps and other geological disturbances occur throughout the field, but the coal lost to date from these causes is less than 1 per cent. of the total. In low-sulfur areas, if the roof coals and their slates are missing and the sandstone replaces them, the sulfur content is abnormally increased. The bottom is either black slate or limestone and, in some places, a thin stratum of fire clay occurs between the coal and the limestone, or the coal and the slate.

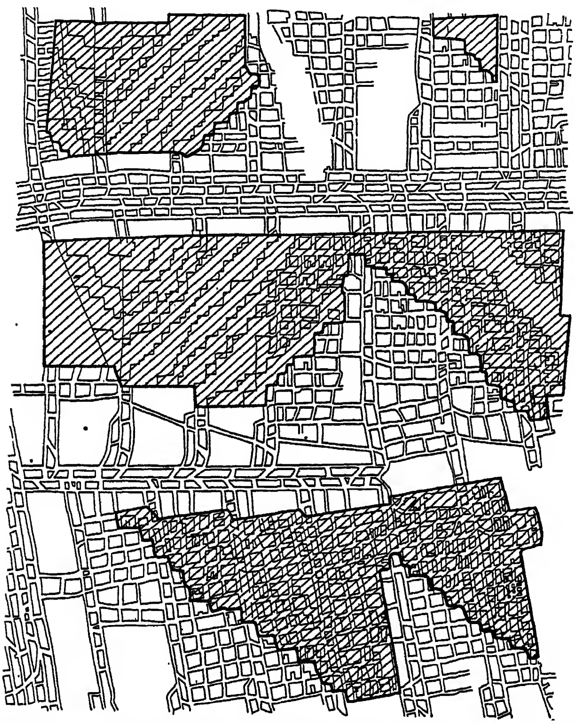


Fig. 6.—Present plan of mining in the Fairmont district, northern West Virginia.

The dips average from less than one-half of one per cent. to as much as 15 per cent. where the bed rises rapidly along the eastern edge of the field. The butt and face cleavage planes, or cleats, are one of the most pronounced features of the coal in the Pittsburgh district. The face course ranges from N. 12° E. to N. 27° E. The hardness of the coal increases from the eastern edge of the district toward the west and the coal decreases in thickness from the east toward the west.

The sulfur in the coal is also higher in the western part of the district and the volatile matter increases progressively from east to west. The

fusing point of ash in the Pittsburgh seam decreases from the northeastern to the southwestern parts of the district. The coal in the same basin, as a general rule, has the same volatile content and, necessarily, uniform fixed carbon. The general dip, aside from that occasioned by the various anticlines traversing the field, is from northeast to southwest.

Thickness of Cover

The maximum cover over the Pittsburgh coal occurs in the southwest corner of Greene county, Pennsylvania, where the coal lies at approximately 1000 ft. under stream level and 1400 ft. under the hilltops. The

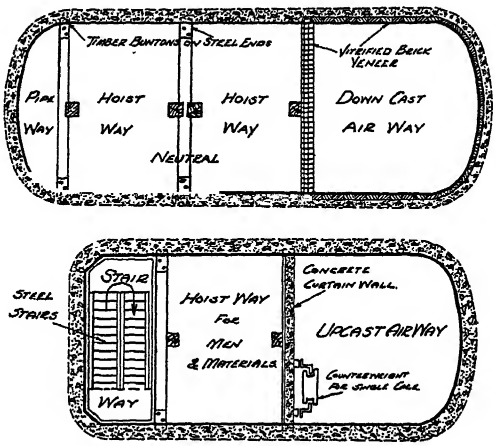


Fig. 7.—Plans of shafts embodying latest practice in design.

cover over the Pittsburgh seam, under the ridges, varies from 1400 to 100 ft., with an average cover of approximately 400 ft. The thinnest cover is naturally along the outcrop and over the isolated patches of the coal occurring on the edges of the field. Coal with as little as 10 ft. of cover is found of good enough quality to serve for high-grade purposes.

Type and Location of Openings

The greatest number of openings in the district, due to the great length of outcrop, are naturally slopes or drifts, but inasmuch as the outcrop coal is being rapidly exhausted the more important openings now are generally shafts or slopes.

The standard shaft today has three compartments, usually about 12 by 24 ft. over-all and 200 to 600 ft. deep. Linings are of concrete, brick, or timber, with steel or timber buntons. The coal contours are already known, or are easily determined beforehand, so that most shafts are located at the low point of the coal area to be worked, in order that drainage and grades for the loads may naturally be toward the shaft bottom.

Hoisting Systems

Where the Pittsburgh seam is too deep to be reached by drift or slope, vertical hoists are used. These hoists are practically all of the balanced type, using platform or self-dumping cages, handling a single car at a time. The older hoists have cylindrical drums, but many of the new hoists are equipped with cylindro-conical drums. Most of the hoists are operated by wound-rotor induction-motors from 150 to 800 hp., that receive power from a central station system.

A few direct-current hoist motors have been installed with flywheel motor-generator set and Ward-Leonard control. This system, although expensive to install, is economical in cost of power, due to low demand, especially at reduced capacity. Skip-hoisting has not been necessary, as the depth of hoist, 150 to 400 ft., has been comparatively small. Present practice in the district is to use electric hoists with steam stand-bys and self-dumping cages.

Underground Mines

The standard track gage underground for the district is 42 in. Rail weights standards are 16 and 20 lb. in rooms; 30 lb. on butt entries and 40 to 80 lb. for main haulways.

Where the coal is drilled by the company, two drillers will average 100 holes in 8 hr. with an air drill driven by a portable electric air-compressor; and, where the coal is undercut an average of 6.5 feet, three holes are used per place, one drill hole yielding approximately 6.5 net tons of coal. Generally, drilling is by hand, using twisted augers making 2-in. diameter holes. Special attention is given by the larger companies in placing of holes and the size of the charge in order that the breakage and the amount of fines may be minimized.

In only a few mines are shearing machines used in the Pittsburgh seam, and although it has been possible to increase the yield in large sizes of coal, the main object in this practice has been to decrease the amount of timbering required and to prevent the handling of the draw slate where it occurs.

Permanent timbering on main haulage ways is usually of steel timber sets, with "H" section beams for legs and "I" sections as cross-bars, and round or square timber sets. The space between the cross-bars and the roof is usually cribbed with condemned posts. In narrow rooms a complete timber set is commonly used for every machine cut. In wide rooms a row of posts is set after each machine cut, on about 4-ft. centers and as close to the face as loading will permit.

Timber is generally delivered to the working places from storage near the tipple in return trips but, in some of the larger operations, timber is delivered to the working places by a night supply-crew.

The minimum size for props is 4 in. at the small end, with length corresponding to the thickness of the seam. Post recovery in pillar drawing is usually about 50 per cent. of the total props used. Approximately 70 per cent. of all the timber used in mining is consumed in the form of posts, caps and cross-bars. The total amount of timber used for all purposes in mining is about 3.8 to 7 board ft. per net ton of coal mined.

One or two mines in the district use treating plants, but the volume of timber treated is so small that the effect is negligible so far as saving in total timber used for mining coal is concerned.

There has been very little actual mechanical loading in Pennsylvania or Ohio. The only practical applications of mechanicalization in large underground areas are in the Fairmont and Scotts Run regions of West Virginia, where both conveyors and shoveling machines have been used on various plans of modified longwall and in room and pillar workings.

Haulage

Locomotive practice varies from gasoline and storage battery types to 35-ton trolley equipment. The mine car is of either wood or steel, 1½ to 4½ net tons capacity. Gathering is done mostly by animals throughout the district, but where height of the coal or severe grades prevent, various types of gathering motors are used. Haulage is done by electric storage-battery or trolley locomotives of various types, gasoline motors, by belt conveyors, and by endless rope. The latest type of underground haulage is the unit conveyor system, used by the H. C. Frick Coke Co. at its Colonial mines in Fayette county. The coal is hauled underground about 23,000 ft. by 20 unit conveyors, the amount handled averaging in excess of 10,000 tons per day.

Equipment

General pumping practice is to use multi-stage centrifugal pumps, usually driven by a 2200-volt a. c. motor. The gathering pumps, most often of the plunger type, are usually portable and mounted on trucks, with capacities varying from 50 to 100 gal. per minute.

The use of air-compressors is limited, for the most part, to small units used in gassy mines near the point of power application.

Practically as many types of mine fans are used in the district as have been marketed in the coal industry in the past 20 years. At different mines fans often handle as much as 400,000 cu. ft. of air per min. against a water gage as high as 3½ in. The general practice is to split the main current of air enough to give each working section a supply that is returned through overcasts to the main return airway without passing through the other workings.

Permanent stoppings are of brick, tile or concrete. Doors are made of wood and steel, and overcasts are brick, concrete, or a combination of both.

Safety Appliances

Watering devices on the cutter-bar of cutting-machines, are supplied in many mines. The working places are often watered before shots are fired, dust being loaded out from entries and main haulage roads. Within the past year, a number of the larger companies have adopted rock-dusting. Frequently the dusting is done on all haulage roads as well as return airways, but there are some mines where rock-dusting is only done on the main haulage roads. This applies generally to the mines in western Pennsylvania and northern West Virginia. Safety lamps are used in mining most of the coal produced in the district.

Labor Used in Mining

Table 1 shows the amounts and kinds of labor used at 27 mines working the Pittsburgh seam in western Pennsylvania and at 2 mines in the thick Freeport seam in the same section, at 33 mines in the Pittsburgh seam in eastern Ohio and at 15 mines in the same seam in the Panhandle of West Virginia. Figures from various mines in the Pittsburgh seam in the Fairmont district show slightly less average amounts of labor per ton of coal, under union conditions, than the average given for the mines in southwestern Pennsylvania. All figures are for the month of greatest production at each of the respective plants.

The data on the amount of labor, in man-hours, required to produce a ton of coal is presented not for the purpose of showing the average amount required by all mines in the district, but to show the performance of some representative plants under both union and non-union conditions. The largest amounts of tonnage labor were used in union mines. The total production is the tonnage produced in the month of largest production during the last 3 years. The thickness of coal represents the respective average thickness of the minable coal section in each mine.

The number of machine men working per day includes both machine runners and helpers. In the figures for the average production per day, the crew of 2 men is used as the divisor. The number of man-hours per ton includes the time of both men and is computed from the actual number of hours in the working day at each mine and not from the time actually spent at the face. All labor required in producing coal is included in the figures, whether employed inside or outside.

In the districts included in the table the total labor cost, expressed in percentage of total mining cost, averages from 70 to 80 per cent. of the total mining cost. The cost of supplies, expressed in the same manner, is from 5 to 8 per cent.

TABLE 1.—*Labor Used in Mining Coal in Pittsburgh District*

		Pittsburgh Seam 27 Mines			Thick Freeport Average (2 Mines)	Pittsburgh & Thick Freeport Seams Total - 29 Mines			Ohio No. 8 District 33 Mines (8 Companies)			Panhandle, West Va. 15 Mines		
		Maximum	Minimum	Average		Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average
Days Worked		27.0	19.9	25.14	22.5	27	18	24.96	26.33	20.94	24.85	27	17	24.6
Total Monthly Production - Net Tons		153,469	12,659	53,027	64,646	153,469	12,659	53,843	69,837	13,197	36,336	43,514	9,774	21,499
Daily Average Production		6,419	487	2,147	2,902	6,417	487	2,199	2,652	630	1,462	1,740	428	874
Average Thickness of Coal		90 in.	53 in.	74 in.	77 in.	90 in.	53 in.	74 in.	64 in.	52 in.	57.5 in.	64.5 in	47.5 in.	55.9 in.
Men Working per Day	Loaders & Pick Miners		51	259.01	274.86	898	51	260.10	1,418	139	643	188	60	98
	Machine Men	78	9.13	14.01	26.22	78	9.13	14.85	153	24	82	25	7	12
	Total Tonnage Men	898	61	273.02	301.08	898	61	274.96	1,556	171	725	213	69	110
	Day Men - Inside	171	30.5	-	96.69	171	30.5	-	482	58	206	88	15	41
	Day Men - Outside	76.4	12.0	-	41.89	76.4	12.0	-	185	24	78	41	10	20
	Day Men - Total	944	54.5	179.26	138.58	944	54.5	176.45	667	82	296	122	25	61
	Total All Men	1,842	115.5	452.85	439.66	1,842	115.5	451.41	2,223	253	1,021	335	94	171
Net Tons per Man per Day	Loaders & Pick Miners	18.92	5.27	8.29	10.56	18.92	5.27	8.45	9.55	7.64	8.49	10.62	7.08	8.91
	Machine Men	125.02	45.71	-	110.67	125.02	45.71	-	94.35	40.44	66.78	109.87	42.50	72.83
	Total Tonnage Men	16.23	4.88	7.66	9.64	16.23	4.88	8.00	8.37	6.79	7.53	9.55	6.07	7.95
	Day Men - Inside	38.49	14.45	-	30.01	38.49	14.45	-	28.78	22.31	26.55	46.57	12.90	21.31
	Day Men - Outside	95.12	23.08	-	69.28	95.12	23.08	-	77.44	36.44	60.16	78.41	26.27	43.70
	Day Men - Total	19.58	6.80	11.98	20.95	21.54	6.80	12.46	20.97	14.24	18.42	27.82	9.03	14.32
	Total All Men	7.13	3.49	4.74	6.60	8.29	3.49	4.87	5.86	4.93	5.34	7.10	3.82	5.11
Man Hours per Net Ton Mined	Loaders & Pick Miners	1.516	0.423	0.965	0.758	1.518	0.423	0.947	1.047	0.838	0.942	1.129	0.754	0.897
	Machine Men	0.175	0.064	-	0.072	0.175	0.054	-	0.198	0.085	0.120	0.188	0.073	0.110
	Total Tonnage Men	1.637	0.493	1.018	0.830	1.637	0.493	1.000	1.178	0.956	1.062	1.216	0.838	1.007
	Day Men - Inside	0.548	0.208	-	0.267	0.548	0.208	-	0.358	0.278	0.301	0.620	0.172	0.375
	Day Men - Outside	0.347	0.084	-	0.115	0.347	0.084	-	0.219	0.103	0.133	0.304	0.102	0.183
	Day Men - Total	1.176	0.409	0.668	0.382	1.176	0.371	0.642	0.562	0.381	0.434	0.885	0.287	0.558
	Total All Men	2.292	1.122	1.688	1.212	2.292	0.965	1.643	1.623	1.365	1.498	1.965	1.126	1.565

Supplies Used

The items represent the principal supplies in amounts usually required for producing a ton of coal:

Explosives, 0.16 lb.	Horsepower:	Pumping, 0.9
Timber, 3.7 board ft.	Coal cutting, 0.16	Ventilation, 1.5
	Haulage, 1.5	Total, 4 kw.h.

Practices in Open-pit Mines

Thickness and character of seam to be stripped and thickness and character of overburden are the most important factors in the operation of open-pit mines. Surface topography, water supply, dip of coal, condition of surface—whether wooded, rocky, barren or improved—improvements on surface, location of nearest railroad, sites for tipples and necessary housing, and availability of electric power, are of secondary importance.

As the No. 8, or Pittsburgh seam, is the only seam stripped in the district, the overburden (with few exceptions) is similar throughout the field. It consists usually of shale, limestone, or shaly sandstone and soil, in the order named from the coal upwards. The topography is much the same throughout; gently sloping hilltops, with strips of varying width between the outcrop and the maximum stripping cover.

Stripping Methods

Deep-mining methods are used where the overburden becomes too heavy to remove and enough coal remains to warrant mining. The usual stripping limits are between the 10 and 50-ft. cover lines, although in a few places coal between the 10-ft. cover line and the outcrop can be used, and coal beyond the 50-ft. cover line is stripped. The principal governing condition is the ability of the shovel to dispose of the overburden at the point of stripping. In going through the crown of a hill, box-cuts are necessary, the stripped material being handled by the coal-haulage equipment. The seam is from 54 to 84 in. thick with the bulk of the tonnage coming from the thinner coal.

Practically all of the coal stripped is used for railroad fuel, general steam purposes, and domestic use, where it gives excellent service. Where picking-tables, shaker-screens and loading-booms are used the stripped product compares in every way with the adjoining deep-mined coal. Sulfur balls of different size are often found in the Ohio No. 8 district. These are laid aside and sold to the sulfuric acid manufacturers. One strip mine in the Pittsburgh district cut across what is known geologically as the Panhandle Trench, a local thickening of the bottom coal. The coal thickened from a normal height of 66 to 120 inches.

The maximum height of cut in the district is 60 ft., with a minimum of 15 and an average of 40 ft. Cuts up to 70 ft. have been made in special cases. The width of the first cut is from 75 to 100 ft., depending upon overburden and spoilage room. The first cut is made along the outcrop of the coal, wherever possible, to the limits of the property, and the excavated material cast to one side. The loading shovel follows immediately behind the large shovel, taking a 30-ft. cut in the coal and leaving a 45-ft. berm for the stripping-shovel on its return cut. Loading is either direct into railroad cars, where conditions permit, or into dump-cars which convey the coal to tipple and picking-tables. The use of picking tables, shaker-screens and loading-booms is practically imperative at present. The return cut of the stripper is made 30 ft. wide, and the material is deposited into the cut made by the loader. This operation is continued to the limits of the stripping.

Machinery

Two types of shovels are used, either operated by steam or electricity, the stripping shovels mounted on double trucks and the loading shovel on caterpillars. The capacity of the stripping shovel is usually 5 to 8 yd, with a 90-ft. boom, a 50-ft. dipper-stick, an operating radius of 125 to 150 ft. and a dumping height of 60 to 70 ft. above the coal bed. The loading shovel has a capacity of 1½ to 2½ yd., a 30-ft. to 40-ft. boom and a 25 to 30-ft. dipper-stick. It is steam-operated.

Dinkey engines of 12 to 50-ton size, and the necessary dump cars, comprise the haulage equipment where the coal is loaded at a tipple. The cars are side or bottom-dumpers, of steel or wood, and from 4 to 16 tons capacity. Gage of track is 36 in. with 40-lb. rails for haulage.

Hard rock (sandstone or limestone), when it occurs in the overburden, is broken by drilling and shooting it for easier removal and to save power and wear and tear on the shovel. The shooting is done well in advance of the work in order to give the elements a chance to aid in the disintegration of the rock. Any well-known type of portable well-drilling machine can be used for this work. Where the coal is hard, it also is shot to to break it up and to save wear and tear on the loading-shovel. Jack-hammer drills are used for drilling the holes for blasting and the charges used are small.

DISCUSSION

W. A. Weldin, Pittsburgh, Pa.—The figures show that the largest amounts of labor paid by the ton were used in the union mines; does that mean there was a correspondingly smaller amount of day labor or a larger number of tons per man?

N. G. Alford, Pittsburgh, Pa.—Generally the tonnage for all men in union mines is less than in non-union mines and in some cases con-

siderably less. As a rule, this condition is accompanied by a relatively low number of tons of coal produced by loaders and tonnage men.

W. A. Weldin.—There is not so much difference, then, in the day men?

N. G. Alford.—A strict interpretation of the union contract usually brings fewer day men on the job.

W. A. Weldin.—Is it necessary to carry more men on the payroll than these figures show?

N. G. Alford.—As a rule in this district, there are from 10 to 15 per cent. of idle men on the payroll.

J. D. Sisler, Harrisburg, Pa.—The United States Coal Commission found that recovery in the Pittsburgh field was approximately 73 per cent. up to and including 1923; the paper does not mention the recovery and coal losses. Has there been any increase in the percentage of recovery in the last 2 or 3 years?

B. Hoffacker, Pittsburgh, Pa.—The percentage of increase in recovery began about 1900. When the first real engineering practices were introduced in the mines, in the Pittsburgh district, up to that time, recovery in parts of Pennsylvania was as low as 60 per cent., and in Ohio was seldom over 50 per cent. and is not over 60 per cent. today. With the adoption of improved practices, the recovery per acre began to rise, especially in the more costly fields like the Westmoreland region and the Irwin gas basin. When the companies began to pay high prices per acre and high royalties for coal, they tried to get more coal out. The people who were receiving royalties demanded that more coal be recovered or that royalties be paid on it anyway. The percentages given in this paper are representative of the coke region, the Westmoreland and gas district, where the coal has sold from $2,000 to $4,000 per acre in the ground.

J. D. Sisler.—Is there a greater recovery in captive mines than in the independent mines?

B. Hoffacker.—The highest recovery might be in the leased mines. A leasing company keeps a closer check on the coal it is getting out than does the ordinary company. All the large companies that have paid high prices for the coal get the maximum recovery; it is above 90 per cent. in most locations.

There has been practically no change in Ohio. Some mines recover about 60 per cent. that probably, in the old days, got 40 per cent. The present percentage of recovery in the No. 8 field in Ohio is slightly more than it ever was. They started in to mine the coal quickly and brought out all of it they could on the advance working and no pillars were drawn.

N. G. Alford.—To a large extent the uniformly low percentage of recovery in the Ohio No. 8 district is largely the result of the thoroughly organized labor conditions there. The fact that the precedents are so thoroughly set and the union contract so well entrenched has no doubt had much to do with adhering to the old plans of working.

J. W. Paul, Pittsburgh, Pa.—In a large part of the Pittsburgh districts the lower bench of coal, the bottom coal, and some of the roof coal were left, an effort having been made merely to get the better quality of coal. During the War, efforts were made to open up a number of coal tracts that had been mined, with the idea of recovering this coal.

When some of these old mines were examined, it was found they were developed on the checkerboard plan, very similar to some of the methods, now coming back into vogue, and mentioned in this paper as being the plan now used in the Fairmont district. The blocks of coal, however, were smaller, usually about 50 ft. square, and the entries were about 10 or 12 ft. wide. Only about 30 per cent. of the coal was recovered in the first mining and no effort was made to recover the pillars.

A number of the plans shown in the paper, some of which are still in use, are unsafe. Where long roof entries are driven, with narrow pillars between the rooms, before these rooms can be advanced to the full extent, the main roof begins to move and the coal begins to crush. That litters up the rooms and the entries and contributes to accidents by obstructing the space along the haulage roads and also disturbs the immediate roof, causing much roof material to fall. It also results in a lessened recovery of coal, as many rooms must be abandoned before they can be driven to their proposed destination and the room pillars are lost.

E. Steidle, Pittsburgh, Pa.—A number of mines in western Pennsylvania are actually, following to the letter the Standard Practices for Rock Dusting Bituminous Coal Mines in the United States, as set down by the Sectional Committee on Standard Practices Institute about a year ago.

C. T. Starr, Pittsburgh, Pa.—How is rock-dusting done in the rooms?

J. W. Paul.—I understand, from people who have visited the mines, that the miners are supplied with the rock dust and paid so much per yard to apply the dust by hand; each day they apply the rock dust on the roof, sides and floor near the face of the room.

F. Dunbar, Pittsburgh, Pa.—Years ago, the Pittsburgh District operated on a 1¼-in. basis, paying the miner for all coal over 1¼ screen. In many cases, the tons produced per acre were low because the miner gobbed the slack—not only gobbed the machine cuttings but a large amount of fine coal. In many cases, this was done upon instructions from the operator.

Nemacolin Mine of The Buckeye Coal Co.

By A. W. Hesse,* Nemacolin, Pa.

(Pittsburgh Meeting, October, 1926)

The trend of American construction toward permanence and longevity is noticeable in the more recent coal plant installations; also, the policy of many coal operators has changed from seeking to obtain the desired tonnage from several plants of small capacities to designing a single installation to produce the whole amount.

The mine openings and plant of the Nemacolin mine of The Buckeye Coal Co., are located on the Greene County side of the Monongahela River, 73 miles south of Pittsburgh. The map (Fig. 1) shows the general location, also the Lambert syncline. This syncline partly accounts for the selection of the site; the other reasons were the possibilities of rail and river shipments.

Two shafts and a slope were sunk to the coal, work starting on all three in the second half of the year 1917. The air shaft was sunk to a depth of 243 ft. and completed in February, 1918; the slope was completed in November, and the skip shaft was stopped at a depth of 21 ft. below the coal, then the concrete lining was stopped 8 ft. above the coal August, 1918. Work on the skip shaft was resumed May, 1923, and sunk to a depth of 60 ft. below the coal by September, but it was not until May, 1924, that the lining was poured.

Opening	Size	Depth or Length	Cu. Yd. Excav.	Cu. Yd. Conc.	Max. Water during Sinking, Gal. per Min.	Min. Water after Grouting, Gal. per Min.
Air shaft	20 ft. diam.	251 ft. 8 in.	4788	1902	475	2
Slope	11 ft. by 19 ft. 1 in.	857 ft.	11590	5200	50	2
Skip shaft	32 ft. 8 in. by 12 ft. 8 in.	296 ft.	6042	2293	300	3

Fig. 2 shows sections of these openings.

The surface plan shows the relative positions of the openings. Coal was first hoisted from the air shaft May 7, 1918, and entry driving was continued; but the coal had to be stored, as the railroad had not been extended from Crucible to Nemacolin. As soon as the slope was driven to the coal and the linings concreted, the entries were driven along projected lines to connect with those advancing from the air shaft, and skip shaft. As soon as the underground connections had been made between the two shafts, the hoisting by buckets was inaugurated at the skip shaft and construction on the fan at the air shaft started.

* Chief coal mining engineer, The Youngstown Sheet & Tube Co.

Slope Bottom

It became evident as soon as entry driving started that the wide spaces would have to be supported to avoid future roof troubles. Experience in neighboring mines presented the probability of not only roof caves but the sluffing of sides above the coal seam. Therefore, plans were made for a double track slope bottom protected by concrete side walls and brick arch. Fig. 3 shows the plan and sections of this bottom.

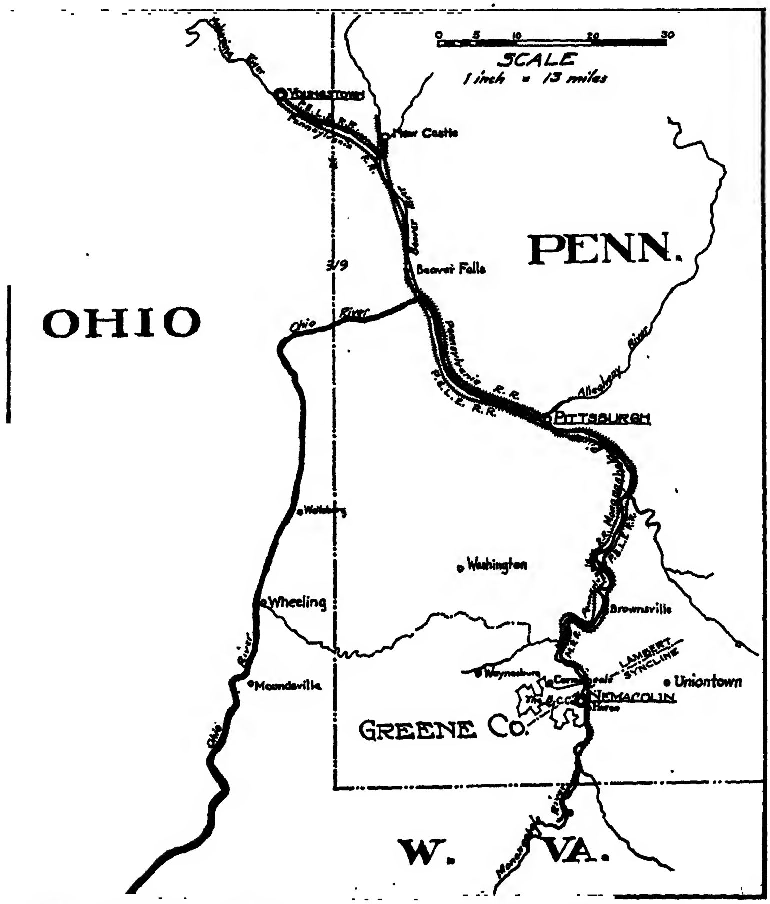


Fig. 1.—Map showing location of coal lands of The Buckeye Coal Co. in Greene Co., Pa.

To avoid congestion around the bottom during the construction, the concrete mixer was placed at the head house, batches mixed outside, dropped into a 6-in. pipe and blown into the forms at the bottom by compressed air. The excavations were made in short sections and immediately cleaned out to avoid delaying the transportation of coal. The forms

were so constructed that cars could pass by or under. Thus, no delays to movement of coal were experienced at any time. This construction required but 5 months, being started March 16, 1919, and completed Sept. 16. The material handled amounted to:

	Cubic Yards
Excavation	5,125
Concrete	838.27
Brick	260,153

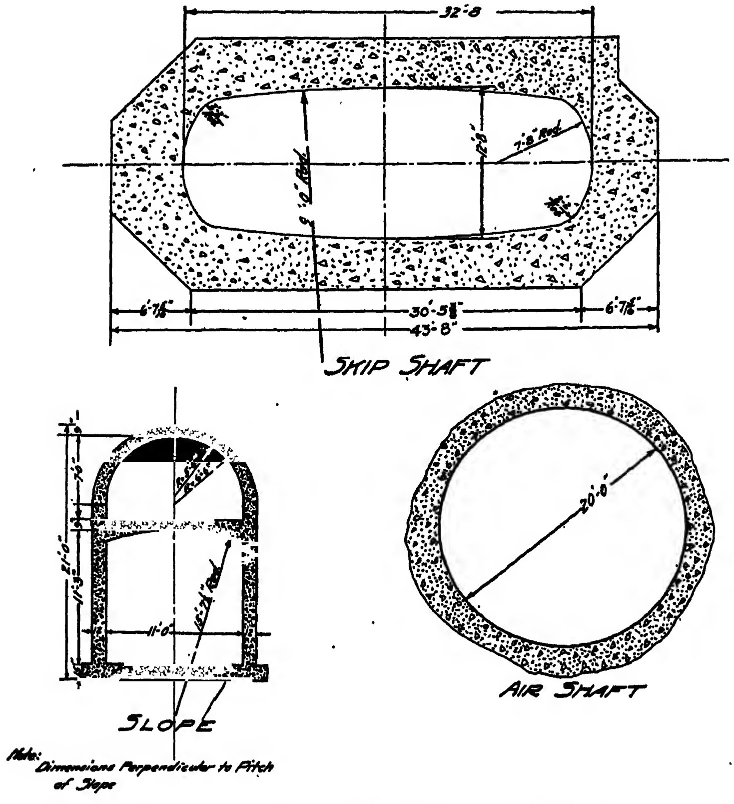


Fig. 2.—Sketch showing section of shafts and slope, Nemacolin Mine of The Buckeye Coal Co.

After the success in blowing concrete through a 6-in. pipe, plans were drawn for a receptacle and blower the result of which is shown by Fig. 4. The concrete blower ordinarily took a batch of concrete containing 0.35 cu. yd. and with 90 lb. air pressure and 3340 cu. ft. of free air per min., the concrete was conveyed to the furthermost point in 3 minutes.

A 12-stall underground stable, concrete lined, is the first job of this blower. The concrete was mixed at the intersection of the manways,

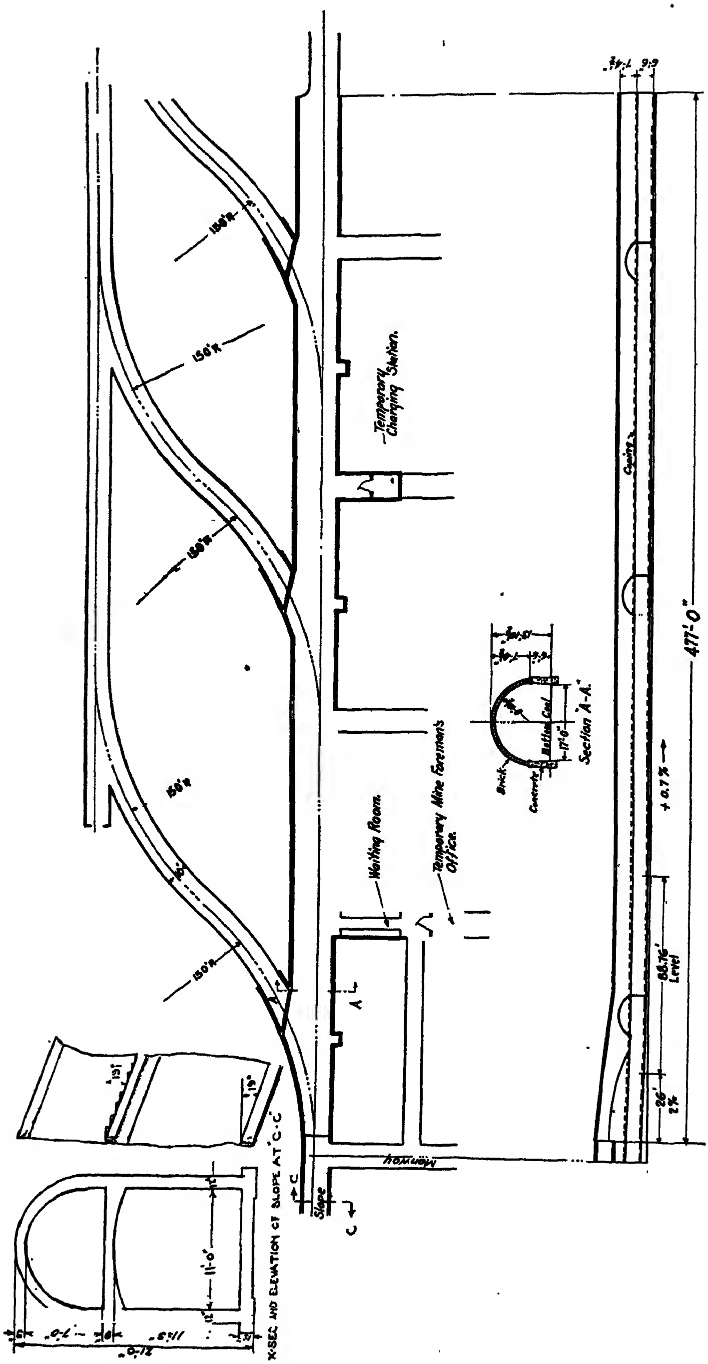


FIG. 3.—SLOPE BOTTOM OF NEMACOLIN MINE.

dropped into the blower and conveyed into the forms everywhere by compressed air.

On Jan. 19, 1919, the connections between the slope and shafts were made, and hoisting from the slope and the skip shaft continued until June, when the mine work was put on a two-shift basis and all the coal was sent out over the slope. With this arrangement the average daily production for 1919 was 500 tons.

FIG. 4.—CONCRETE RECEPTACLE AND BLOWER.

OVERHEAD CROSSOVERS AND STABLE EXTENSION

An unusual feature of this mine is the method used in reaching the slope bottom from the four main line tracks to the skip shaft bottom. Grade crossings were eliminated; and where the trips from the north and south sections run into the "loaded crossover" an open space has been provided, which the employees call the "picture show," to give the motormen a clear view and prevent wrecks. This is all brick and steel

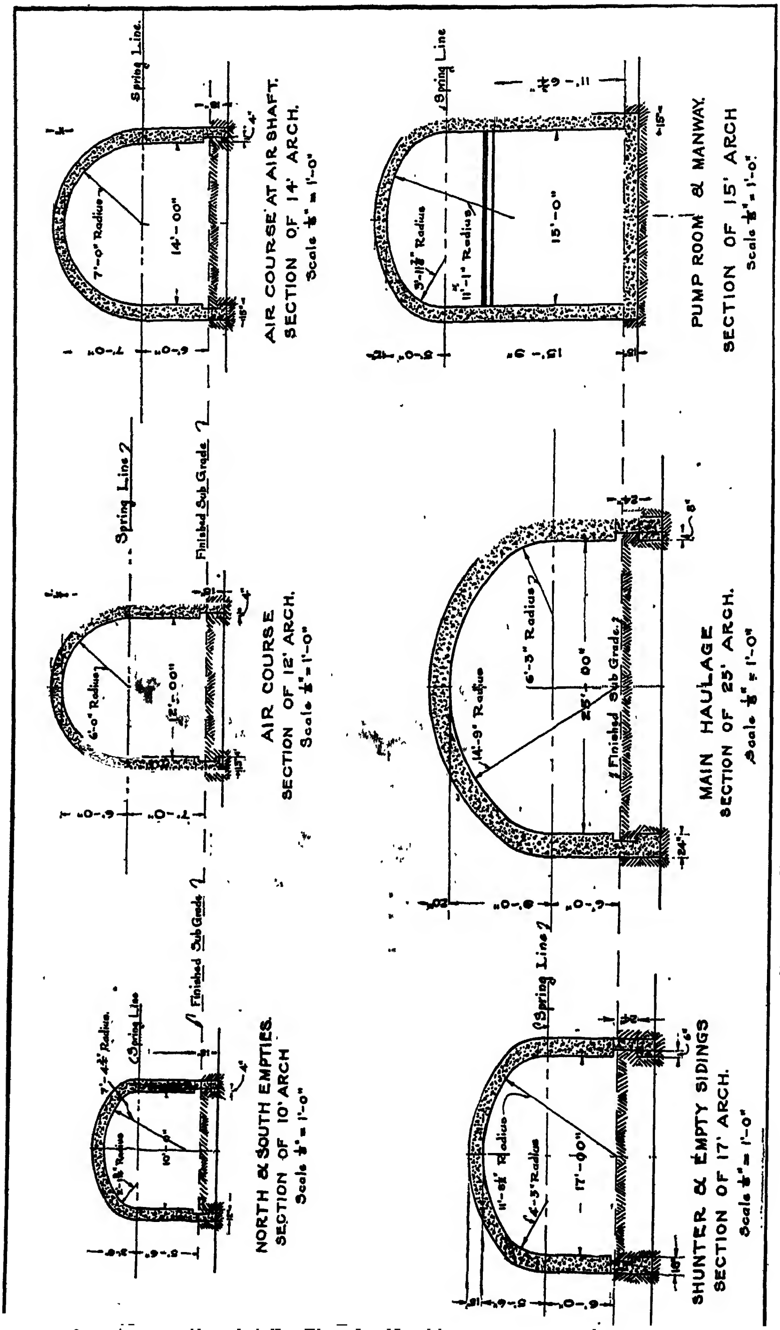


FIG. 5.—CROSS-SECTIONS OF CONCRETE ARCHES AT MINE BOTTOM.

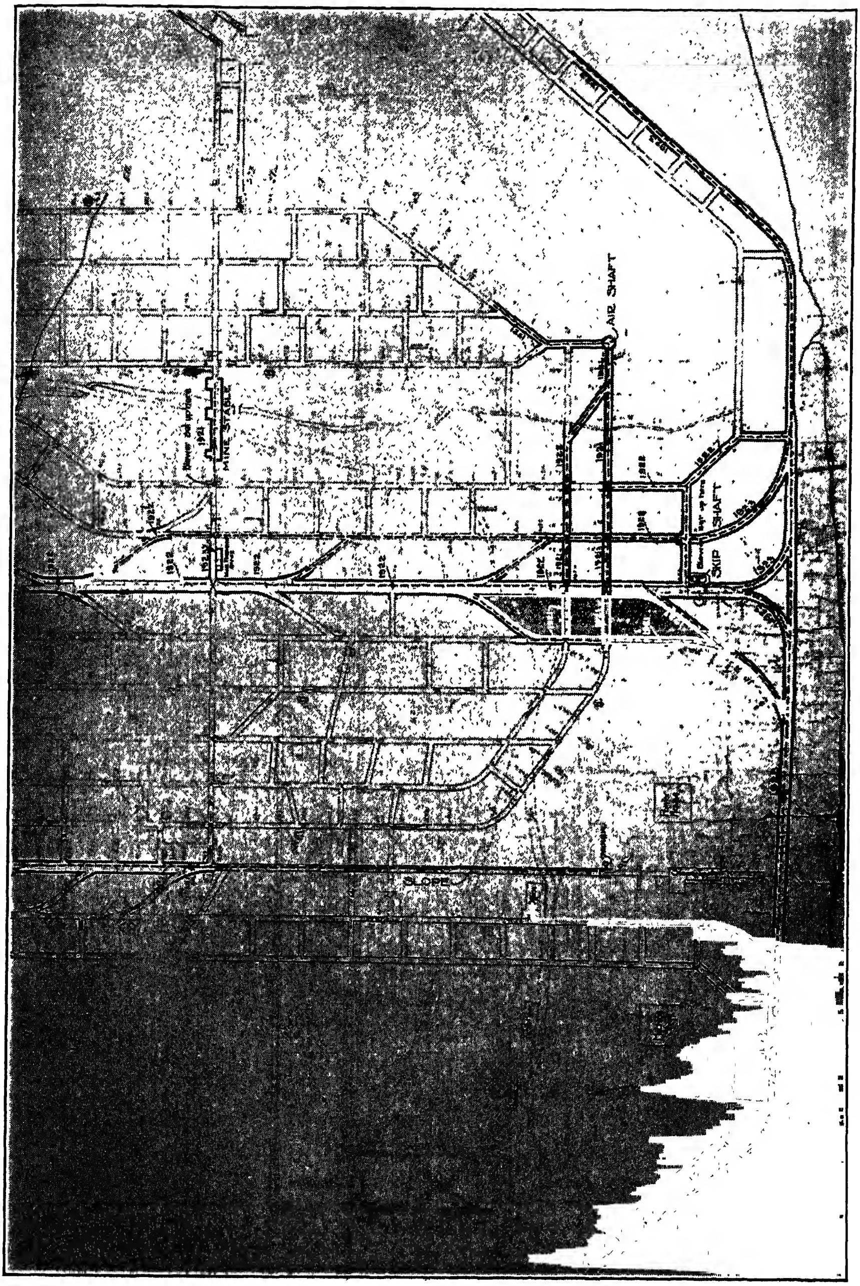


FIG. 6.—CONCRETE CONVEYOR LINES. (SCALE: ½ IN. = 100 FT., APPROX.)

and the work was performed without stopping the flow of coal past this point. About 1800 tons per day were then being produced and the average for the year 1921 was 1545 tons.

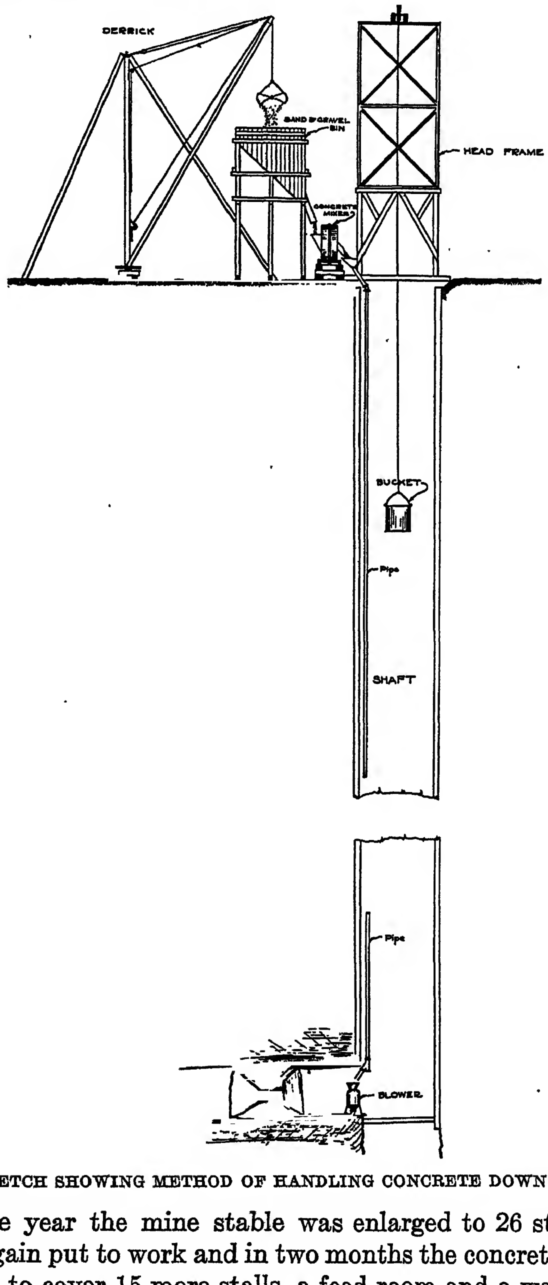


FIG. 7.—SKETCH SHOWING METHOD OF HANDLING CONCRETE DOWN MAIN SHAFT.

This same year the mine stable was enlarged to 26 stalls, and the blower was again put to work and in two months the concrete lined stable was extended to cover 15 more stalls, a feed room and a wash room.

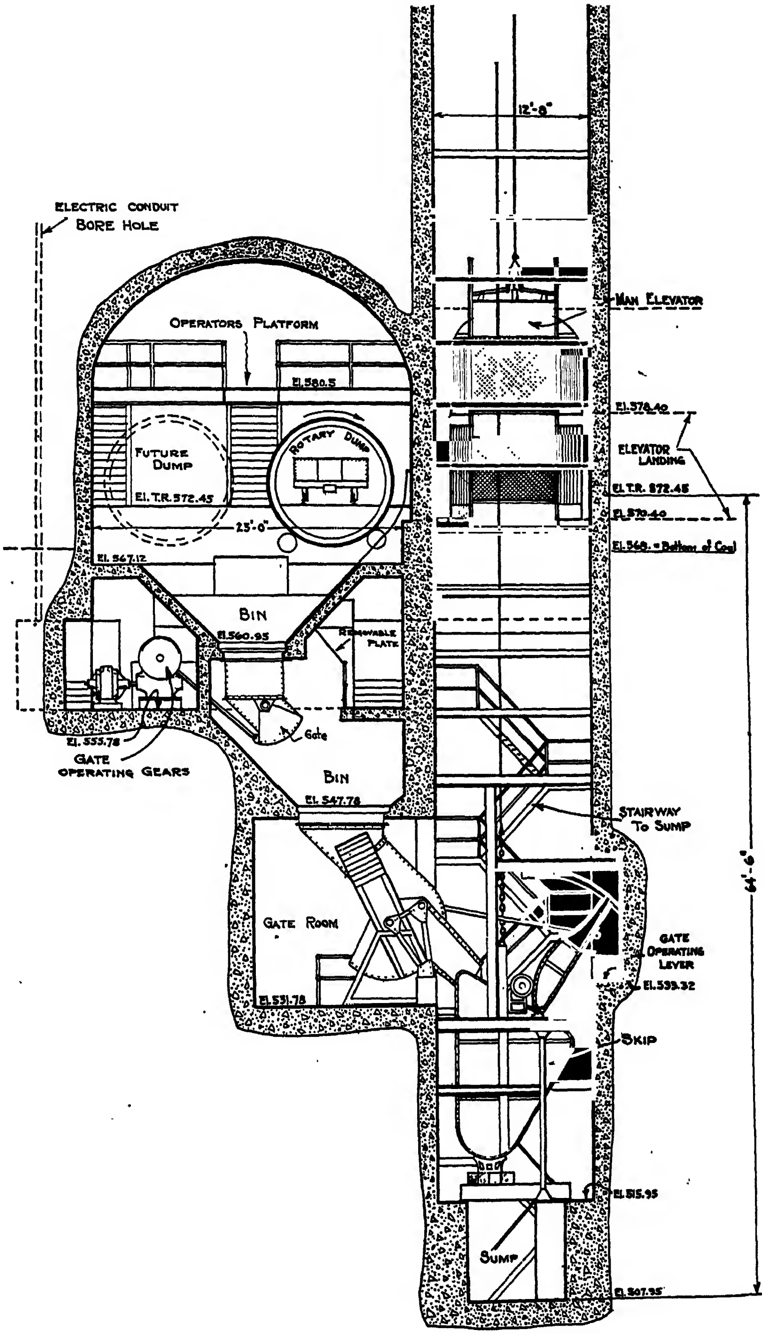


FIG. 8.—SECTION THROUGH MAIN SHAFT BOTTOM OF NEMACOLIN MINE.

The slope was designed to handle supplies, air and slate and provide for man travel, so when put into service for coal haulage, it was not

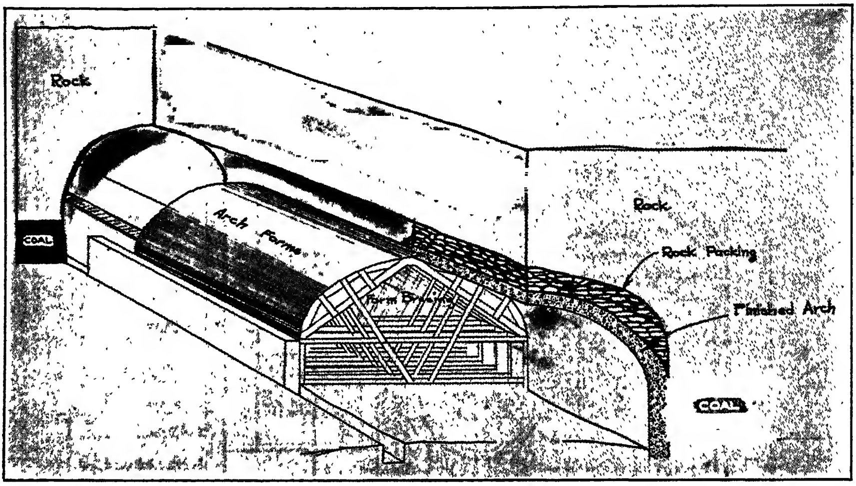


FIG. 9.—ARCH FORMS, CONCRETING, PACKING, ETC.

expected to go over 2200 tons per day; but the development of the entries in 1920 indicated that it would be necessary to start on the shaft bottom immediately to take the increased production.

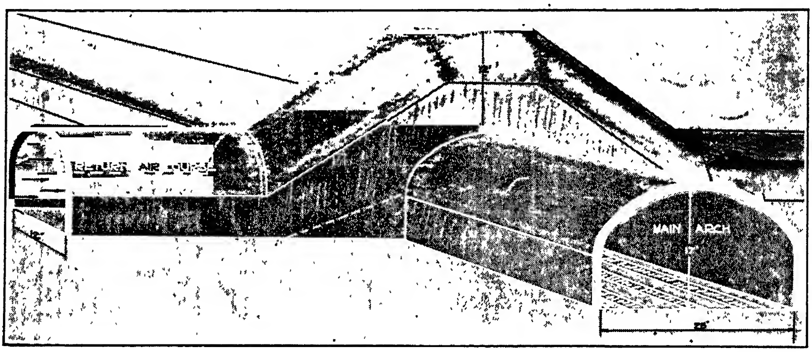


FIG. 10.—ARCHES AT INTERSECTION OF MAIN LOADED TERMINAL AND NO. 3 SOUTH AIR COURSE.

CONCRETING MINE ENTRIES AROUND SKIP SHAFT

The plans for concreting were drawn and the work started March 4, 1921, in the air courses at the bottom of the air shaft. Fig. 5 shows the finished cross-sections of the various entries involved; Fig. 6 shows the

ramifications of the concrete conveyor pipe during this construction. None of the concrete was placed by hand. A derrick, on the outside unloaded the railroad cars of sand and gravel and loaded the bins. The

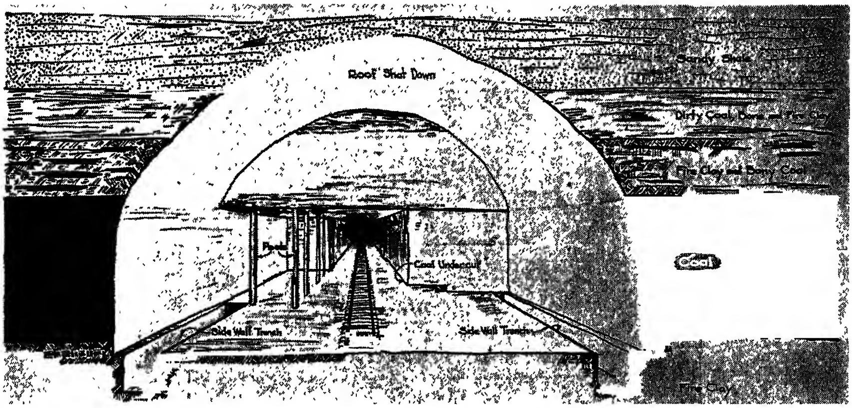


FIG. 11.—SKETCH SHOWING PROCESS OF EXCAVATION, MAIN LOADED TERMINAL.

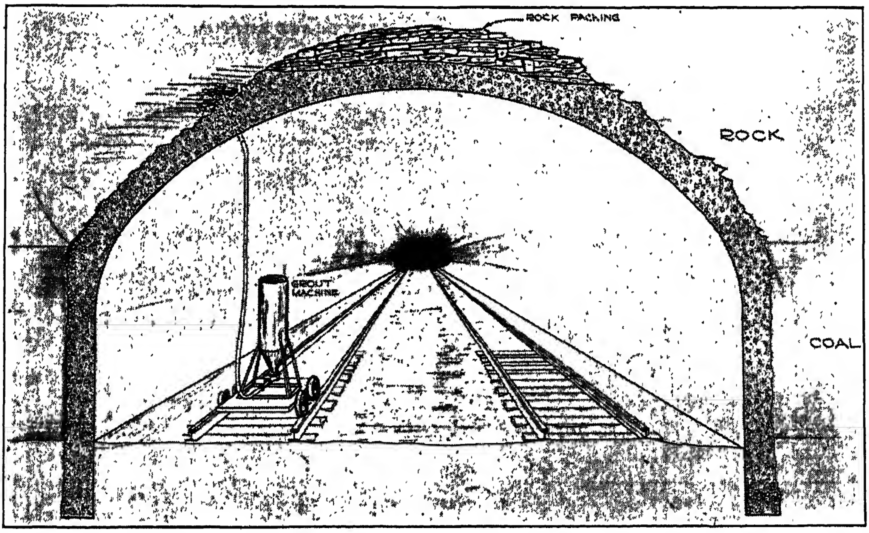


FIG. 12.—SECTION OF MAIN ARCH, SHOWING ROCK PACKING AND GROUTING.

sand and the gravel bins stood above and a little to the rear of the mixer, permitting the flow of aggregates to measuring chutes and then to the mixer at shaft top; from the mixer a 6-in. pipe carried the concrete to the bottom and into the blower; from this point the concrete was sent to every

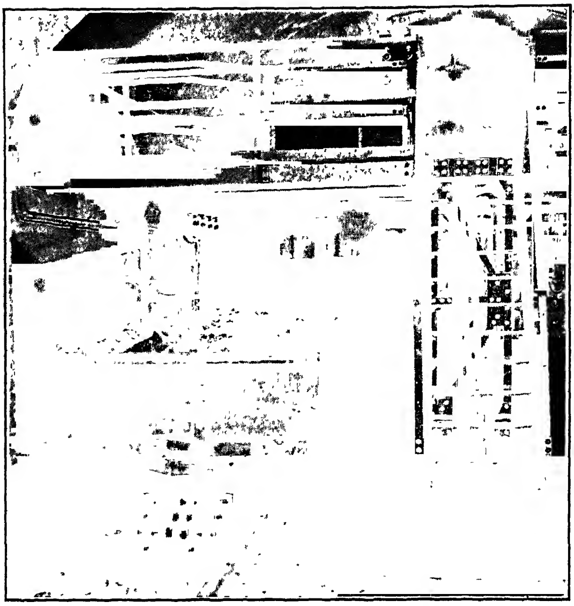


FIG. 13.—ROTARY DUMPS, MAIN SHAFT BOTTOM.

FIG. 14.—MAIN HAULAGE TERMINAL.

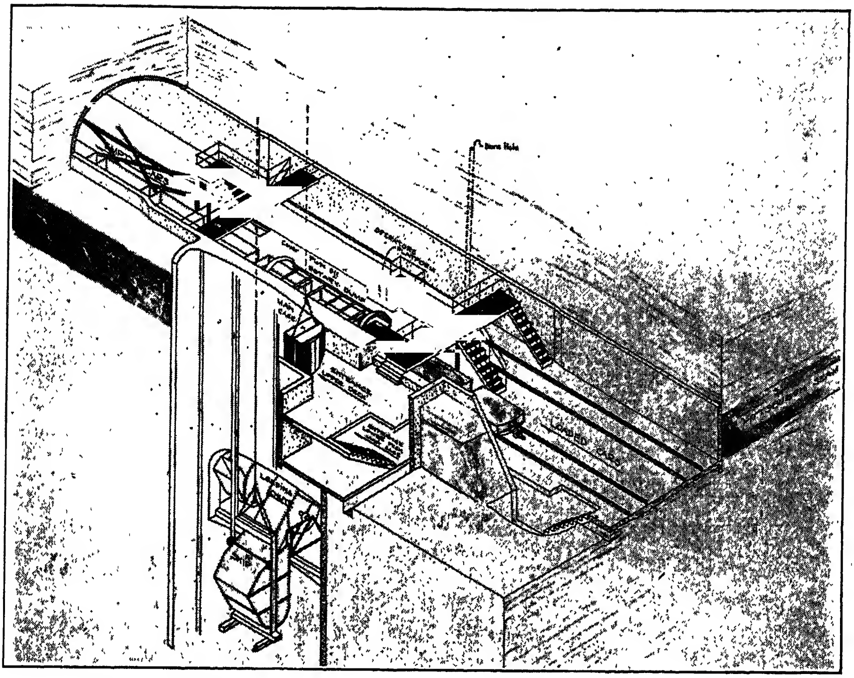


FIG. 15.—SKETCH OF MINE BOTTOM SHOWING SKIP AND MAN CAGE.

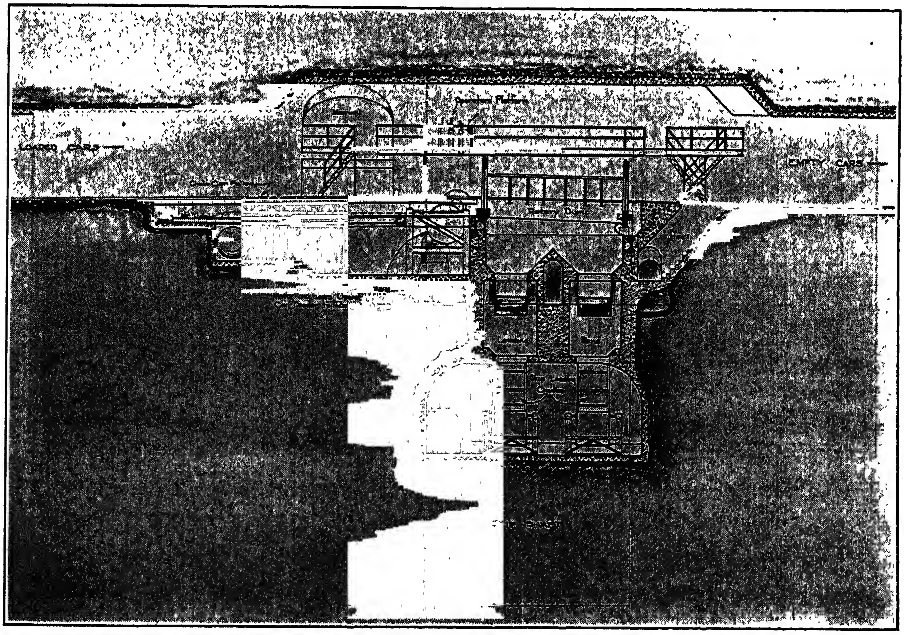


FIG. 16.—LONGITUDINAL SECTION THROUGH MAIN SHAFT BOTTOM.

point of construction through 6-in. pipes. Fig. 7 shows the outside arrangements.

Between March 4, 1921, and July, 1925, there were handled 63,880 cu. yd. of excavation and 27,837 cu. yd. of concrete, but the same organ-

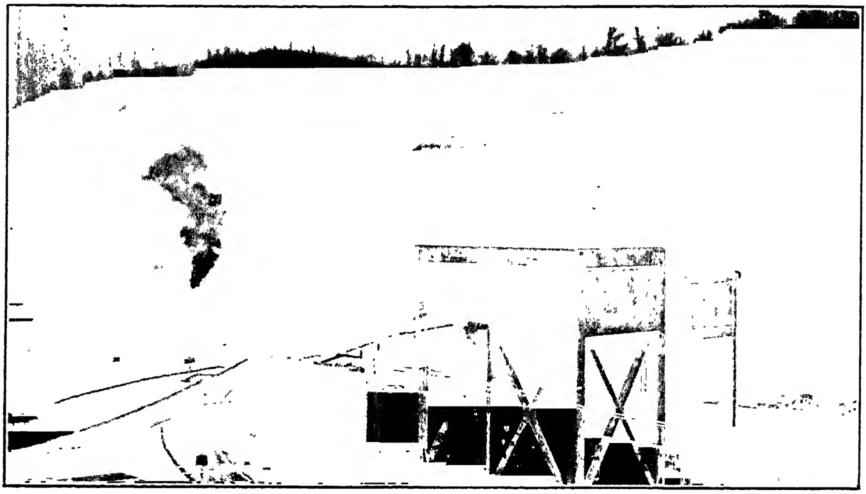


FIG. 17.—SLOPE TIPPLE AT NEMACOLIN MINE.

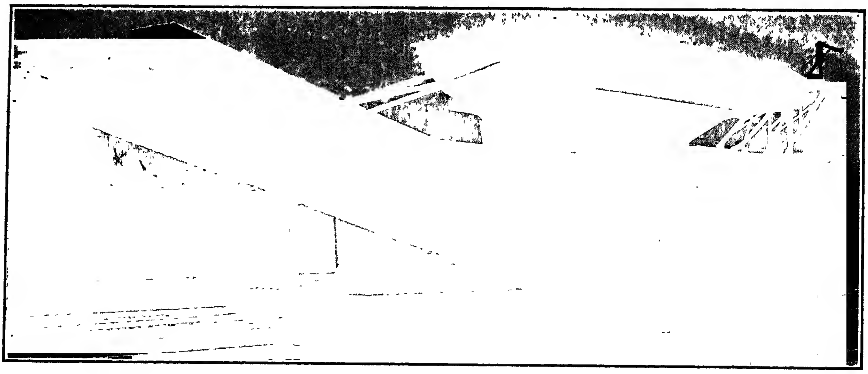


FIG. 18.—TWENTY-TON ELECTRICALLY-DRIVEN SLATE LARRY.

ization also constructed the stable, mine foreman's office, overcasts in that vicinity, and shaft and bins, increasing the quantity of material handled to 68,010 cu. yd. of excavation and 30,303 cu. yd. of concrete.

The progress of this concrete arching by years is shown by the following table:

Size	Feet of Entry Concreted 1921	1922	1923	1924	Total
25-ft. arch	263	952	190		1405
17-ft. arch			792	356	1148
15-ft. arch pump room		63	19		82
14-ft. arch	180				180
12-ft. arch	778	242			1020
10-ft. arch	729	498	1494	674	3395
Totals	1950	*1755	2495	1030	7230

* Work delayed by nation wide strike.

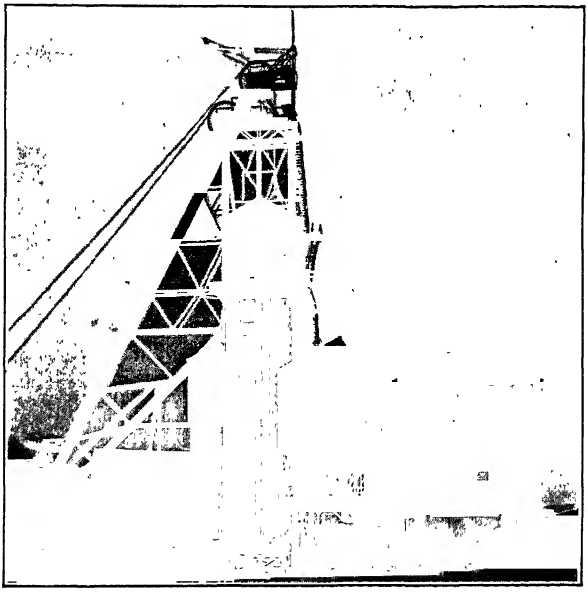


Fig. 19.—Main tipple at Nemacolin Mine.

Fig. 20.—Picking tables at Nemacolin Mine.

Figs. 8 to 16, inclusive, show views of the underground construction. All concrete work in the mine is reinforced.

The rotary dump movements are controlled manually and the flow of coal to the skips automatically. As the skip leaves the bottom (Fig. 8) it closes the discharge gates of the lower bin. As soon as the gates are closed, the skip is passing a control contact, opens the upper bin gates just long enough to fill the lower bin to a 12-ton measure. When the skip approaches the bottom again it engages the sword arm of the gate lever and in landing pulls the lower bin gate open to discharge its load.

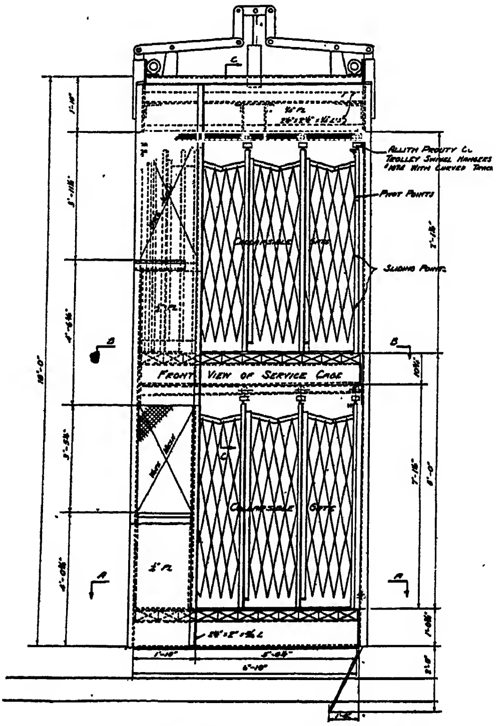


Fig. 21.—Man cage in main shaft.

Comparatively light forms (Fig. 9) were used for the concrete; the arch centers were used three or four times before becoming worn out. Rock packing was often started 24 hr. after pouring the concrete and many forms were removed within 48 hr. All concrete was 1:2:4 mix.

The main arch (Fig. 10) was built before all the excavation was made for the arch to go over the top.

The excavation (Fig. 11) was made with hazards of rock falls reduced to a minimum. Not an accident from this source occurred during the entire 3½ yr. on this work.

The sealing-off of the water was eminently successful, considering the packed spaces over the concrete (Fig. 12). In places, flows of 50 to 100 gal. per min. were completely shut off.

The skips compartment (Fig. 15) is sealed off from the bins and dump, the lower gates being closed when the upper gates are operating and vice versa.

This longitudinal section (Fig. 16) shows the position of the operator in relation to the locations of the chain car haul, the measuring bin gates and the discharge gates. The operator can see the movement of the cars and the dump, but the other units are out of view; their positions are indicated by signal lights on a board before the operator.

Tipples

The slope tipple of reinforced concrete and steel was erected in the latter part of 1918. One feature of this structure is the location of the 400-

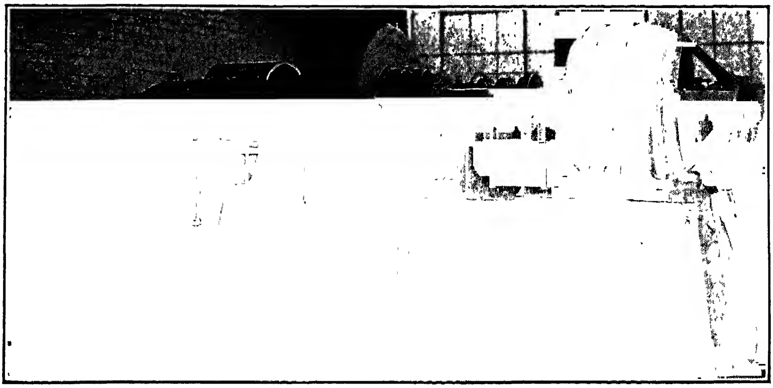


Fig. 22.—Fan reduction gear at Nemacolin Mine.

hp. hoist above the haulage track on the tipple; see Fig. 17. Two single-car rotary dumps in tandem are used to turn the cars over, thus permitting coal to be dropped into one bin and slate in the other. The coal may be shipped by railroad from this point or by truck for domestic use. The slate is taken away by a 20-ton electrically driven slate larry; Fig. 18.

Construction of the main tipple was not started until March 11, 1924; 845 tons of structural steel were used. The bin that takes the coal from the skips has a normal capacity of 168 tons. From here, the coal is fed onto two gravity bar screens by parallel conveyor feeders; the lump coal going to pan picking tables and the egg is separated from the screenings by shaking screens, the resultant sizes being carried parallel to the lump coal on belt conveyors. At the ends of these conveyors, a mixing conveyor traveling at right angles to the picking tables, takes the coal to a small bin from which it is fed into railroad cars. Thus, picked, lump

and egg, lump and slack, egg and slack, with slack, egg and lump, respectively, or picked run of mine, may be shipped. Exterior and interior views are shown in Figs. 19 and 20. The tipple was put into operation May, 1925, and has a capacity of 1500 tons per hour.

The hoist house at the shaft contains two bin and picking tables, flywheel motor-generator sets, one 1400-hp. hoist and one 175-hp. man elevator hoist, besides all the necessary accessories. The man elevator

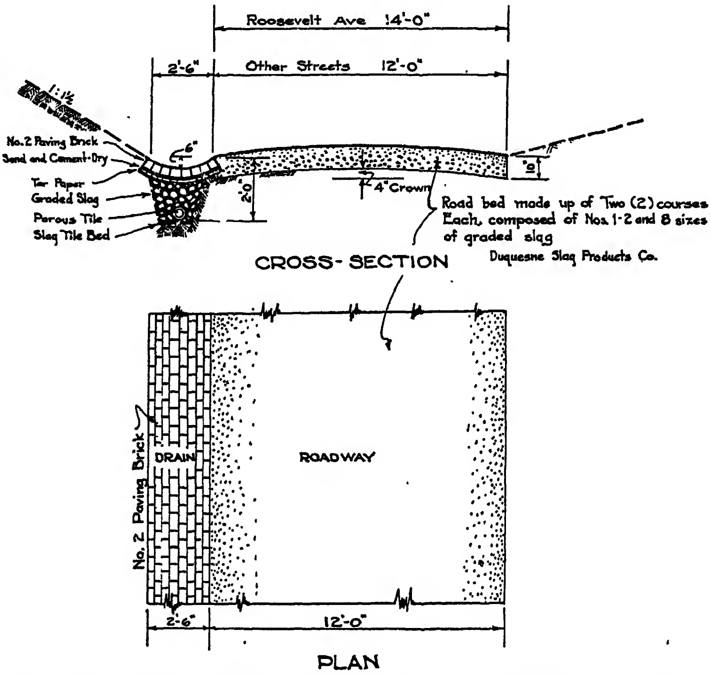


FIG. 23.—TYPICAL PLAN AND CROSS-SECTION OF NEMACOLIN ROADS.

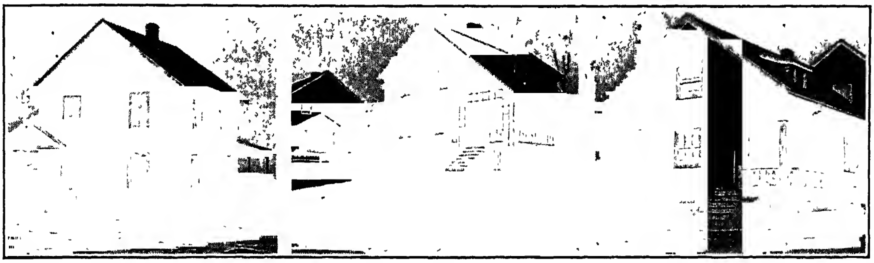
is a double-deck cage in which 16 men can ride on each deck, or 32 men in all. It is estimated that 1500 men can be lowered or hoisted in 45 min.; see Fig. 21. All the wiring between the power house, hoist house, main tipple and mine fan is carried underground, in fiber duct enclosed by concrete.

MINE FAN

A 14 ft. by 6 ft. Jeffrey multiblade fan furnishes the ventilation for the mine. This fan, as well as everything else about the plant, mine and town, is operated on purchased power, though a generator driven by two GRC 8 Sterling gas engines is provided to keep the fan moving to full speed in an emergency. There are two drives on this fan—one by reduction gear and the other by belt—both being operated by 400-hp. motors. Fig. 22 shows the fan reduction gear drive.

Railroad Sidings

The railroad sidings have a capacity of 120 empty and 120 loaded cars. The empty tracks are laid with 80-lb. rail and the loaded with 100-lb. rail. The empty and loaded tracks are level, except where the loads leave the tipple to a point 500 ft. below. A set of railroad scales, with a Streeter-Amet weighing device, weighs the coal about 150 ft. below the center line of the tipple. The empty cars are pushed down to a point where the railroad car puller can reach them; this puller is a double drum hoist set in a concrete pit about 320 ft. above the tipple. A continuous cable extends from the hoist to a sheave at the lower side of the tipple. Short pieces of cable with a hook on one end and a grip on the other serve



(*a*) (*b*) (*c*)

Fig. 24.—Dwellings at Nemacolin, Pa. (*a*) 6-room dwelling. (*b*) and (*c*), 5-room dwellings.

to connect the trip of railroad cars to the continuous cable. A 50-ton steam locomotive shunts the cars around the yard.

Machine Shop

The machine shop is built of steel and brick, 105 ft. wide and 80 ft. long, with provisions made to lengthen it 120 ft. more. All classes of mine equipment repair work are done in this building.

Town of Nemacolin

The streets were all laid out with grades not to exceed 10 per cent. This was done in every case, except one connection from the temporary store to the theater. Four types of construction were used: the brick road and the water-bound macadam road are shown in Fig. 23; the other two types are a sandstone Telford base with graded slag and tarvia binder on top and a sandstone base covered with granulated slag. The brick and tarvia roads were completed June, 1921. Since then, very little patching has been done on the tarvia, while the brick road has

needed no attention, except to fill expansion cracks. Drainage has been well taken care of, to which fact can be ascribed longer lives of the road.

There are twenty-five types of houses and these types have slight modifications to break up the sameness of appearance. Figs. 24 and 25, incl., illustrate the outside features. Seventy-three of the houses have concrete foundations, sixty-two tile block, and the remainder concrete

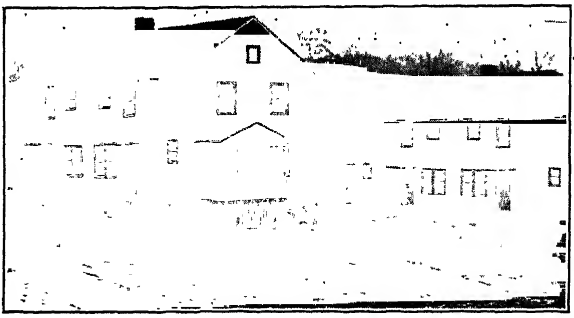


Fig. 25.—Two-family dwelling at Nemacolin, Pa.

block plastered on the outside. All the houses have concrete basements, which contain furnaces, coal bins and laundry space. Bath tubs or showers provide the facilities for cleanliness, as every house has a hot water tank and piping, in addition to the cold-water facilities. Sanitary sewers take care of the waste water and toilets. The sewage is carried

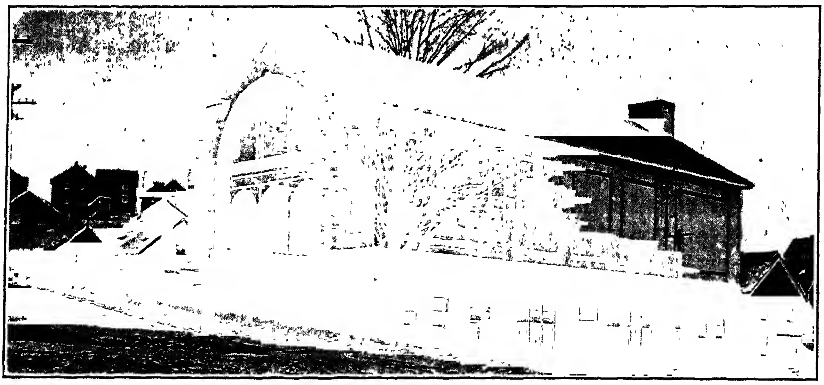


Fig. 26.—Theater at Nemacolin, Pa.

to two Imhoff tanks, just north of the hoist house, and the effluent is treated with chlorine. The streets and houses are lighted by electricity. The theater is shown in Fig. 26. No water mains throughout the town are smaller than 6 in. in diameter and pressure is obtained by a 20-ft. diameter by 110-ft. high standpipe set on the highest point.

Operation of Nemacolin Mine

By W. Z. Price,* Nemacolin, Pa.

(Pittsburgh Meeting, October, 1926)

The coal lands that the Nemacolin mine is to develop embrace over 8400 acres; the tract is oblong and its eastern edge is along the Monongahela river. As shown in Fig. 1, the mine is divided into two parts (the North mine and the South mine) by a 150-ft. barrier, which however, is cut through at intervals by haulageways. Each part is fed from seven main entries—a loaded track, an empty, a manway, and four return air

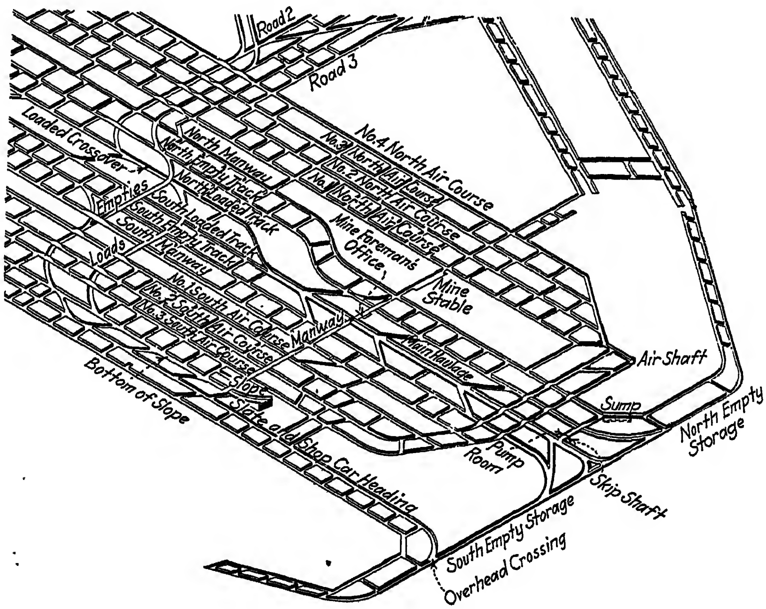


Fig. 1.—Bottom layout of Nemacolin Mine of The Buckeye Coal Co.

courses. The loaded tracks converge at the shaft bottom but the empty roads maintain their identity throughout their length. With three intakes and four returns on either side, there is ample room for expansion of air and every facility for ventilating is provided.

The four south air courses converge into two, crossing the shaft bottom, and then into one at the south side of the air shaft. The four north air courses converge in a similar manner on the west side of the same shaft. This convergence is accomplished by concrete arched

* Superintendent.

entries, which increase in size as additional air is supplied them. A minimum of 20,000 cu. ft. of air per min. passes through each split and more where required; this is not intake air, but the air through the last open crosscut in each section. Twelve such splits are in circulation and two must be added before the end of this year. The mine is gaseous and thoroughly electrified, therefore ample air is provided for both safety and mining efficiency. The quantity of air is frequently checked by the mine foreman, his assistants, and the safety inspector. Permanent brick walls have been erected in the crosscuts between the intake and the return in all splits. These walls are a single brick thick and have a pilaster 1 ft. square in the center. This is both economical and sound construction. Overcasts are built with I-beams and jack arches for the floors and brick walls for abutments.

All working places in virgin coal are ventilated by line brattice. The area behind the brattice is held to a minimum of 12 sq. ft. and the volume to at least 2000 cu. ft. per min. Temporary brick stoppings in room headings are laid with lime mortar and all joints are made as air-tight as possible. On all room headings, the air is checked with curtains of coarse, non-inflammable, jute cloth.

Should the fan stop for any reason, a gong in the power house is rung automatically; the power house engineer starts the fan with the other drive, if the first has failed. In the case of failure of the West Penn power supply, he immediately cuts off all power to the mine, notifies the mine foreman, then starts the two 300-hp. gas engines; these are directly connected to a 400-hp. generator wired to the fan motors and the ventilation is maintained. He then switches the fan over to the other West Penn line and stops the gas engines; the plant is then operated as before. The emergency unit is tested daily and is kept in perfect condition. A private telephone line to the underground exchange enables the engineer to communicate promptly with the mine foreman at all times. A card on the switchboard constantly reminds the engineer of the proper procedure in such instances.

Mining

Coal is cut by both shortwall and top cutting mining machines. The later are used for development work while most of the butt and all room work are done by the former. Rooms and butts are driven on 100-ft. centers; the method of drawing the pillars is shown in Fig. 2. Rooms are not driven until they are ready to be brought back; except when they are needed for haulage or ventilation. By this system practically all the coal is recovered with maximum safety. The coal is cut, then drilled and shot down by company men. The drillers work in pairs, like the cutters, and use a portable air compressor, self-propelling, thoroughly explosion proof and approved by the Bureau of

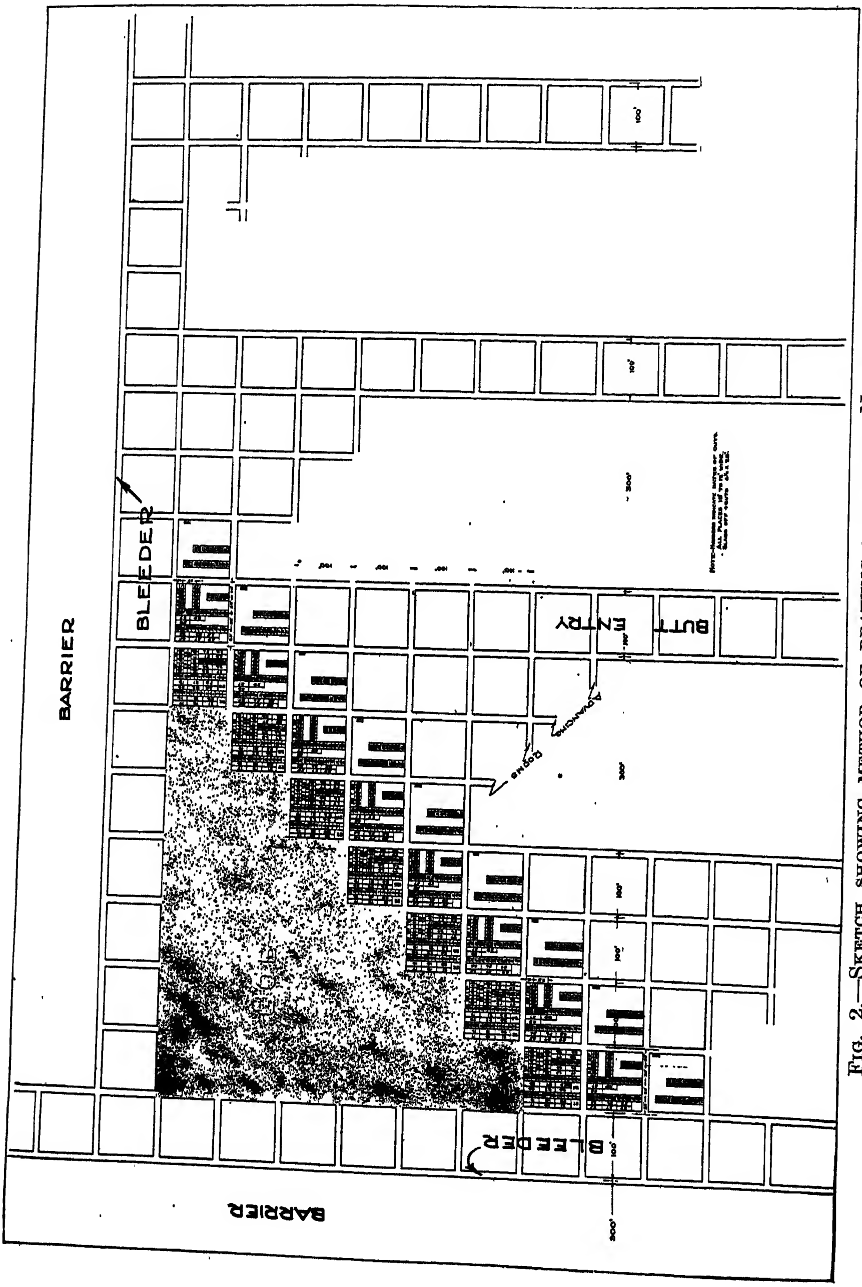


FIG. 2.—SKETCH SHOWING METHOD OF DRAWING PILLARS AT NEMACOLIN MINE.

Mines. Rotating air drills with augers are used and the three holes are drilled in 5 or 6 min. The shot firer and his helper shoot the coal by means of permissible explosives and electric detonators. The explosives are taken underground in a car, Fig. 3, especially built for the purpose, the body being doubly insulated from the running gear and hitchings and mounted on a truck by spring suspension. Each loader loads his complete cut each day. After loading his cut he extends the track to the face for the top cutting machines, and the cycle at the face is complete. With the shortwall machines, the loader lays his track the first thing in the morning.

Few haulage roads are driven with intersecting angles, as curves are simple to drive and conform more easily to trackwork. The engineering department furnishes the underground management with sketches of the curves that are to be driven and full details for the continuance of the sight lines. No place or crosscuts are turned until sights are given by the engineers. Crosscuts are driven at regular 100-ft. intervals in most cases.

In shot firing, the places are first examined to see if they are well sprinkled and free from gas, and if the holes have been properly placed. After shooting, the shotfirer tests for gas before proceeding to the next place. All electric blasting batteries are checked in and out of the

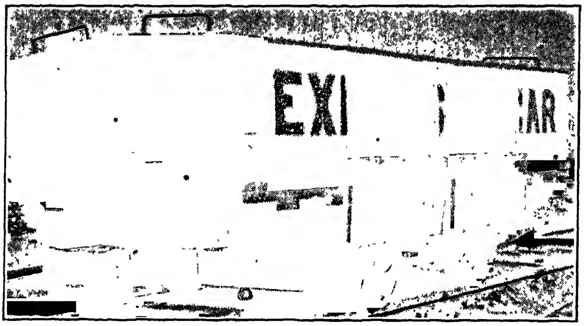


Fig. 3.—Car in which explosives are taken underground.

lamp house by the lampman in charge and no one except the shot firers, or persons designated by the mine foreman, can have possession of one.

Both the shortwall and the top cutting mining machines are equipped with water sprays on the cutter bars. A ¼-in. pipe is fastened to the frame of the machine. This tapers to a jet that feeds on the bits, in the case of top cutters; but on the shortwall machines, a horizontal perforated pipe throws a spray across the whole cutter bar. About 0.7 gal. of water is used per ton of coal cut.

Rock dusting is practiced on a large scale. Although but few barriers have been erected, all places where track is present have been

thoroughly dusted. In advancing air courses, the roof and ribs are coated for a considerable distance where the track has been removed; certainly far enough to suffice for scores of barriers. Two machines are used and the dust is blown on at the rate of 4 lb. per ft. of entry or room (all are 12 ft. wide). The dust is sampled periodically and the percentages of incombustibles are indicated on a map kept for this purpose.

Haulage

Three sizes of rail are in use. The 70 lb. for the main haulage roads, 50 lb. for secondary tracks, and 30 lb. for the butts and rooms. The first two sizes are classed as permanent track and are put down on grade and line with 6 by 8-in. ties at 2-ft. centers. The light rail, of course, is being constantly changed and is laid on 4 by 6-in. ties. The 30-lb. rail is bought in both 30 and 15-ft. lengths. Standard turnouts are prescribed for varying conditions; these are constructed under the supervision of the engineering department.

All car movements are regulated from the dispatcher's office underground. All roads in the mine are numbered; this facilitates dispatching and minimizes confusion in the work and in correspondence. Records are kept of the number of cars entering or leaving each station, which is the section sidetrack or the shaft bottom. This system has 3 advantages: It enables the mine foreman or transportation boss to get a "bird's eye" view of the production of all or any part of the mine at any time during the day; it reduces transportation delays; it systematizes the transportation. A dispatching sheet is shown in Fig. 4.

A dispatcher is at the post throughout the day. Each section foreman reports to him the number of cars that he will require that day; that information is placed at the top of each sectional car record, together with the number of cars on hand, both loaded and empty. No motorman enters or leaves any side track without reporting to the dispatcher and receiving instructions regarding the disposition of his cars. To maintain this system requires an elaborate telephone installation. One 50-pair, lead-covered, telephone cable connects the dispatcher's switchboard and the mine workings; the individual wires are strung on the rib near the working places. All cables are laid in 6-in. trenches filled with lean concrete.

Coal is gathered by storage-battery and cable-reel locomotives. The latter are 9-ton, explosion-proof, haulage units. A 400-ft. concentric cable connects with the trolley wire out in the fresh air. The concentric or double cable provides a positive return to the outer or bonded track so that it is used for the mining machines and portable air compressors as well. Main haulage is with 13-ton trolley locomotives of standard

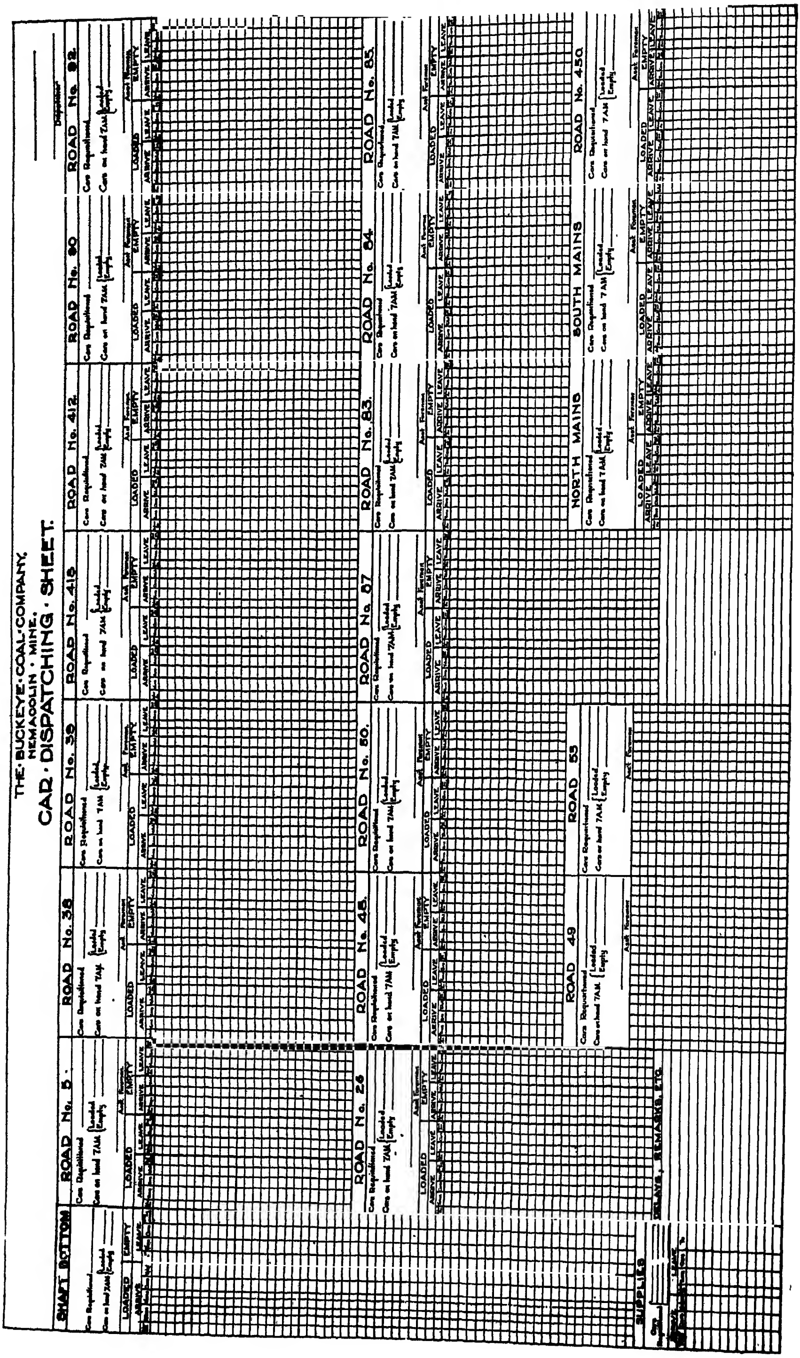


FIG. 4.—DISPATCHING SHEET EMPLOYED BY THE BUCKEYE COAL CO.

design, the track is 44-in. gage. The cars hold approximately 3 tons and are 6 by 10 ft. in area and only 36 in. high (above rail). They are solid cars as revolving dumps are used; this tends toward clean haulage roads. These locomotives are so equipped that they can be operated in tandem when desired.

On straight track, the trolley wire is supported every 20 ft. when maintained at normal height; when conditions require hanging below that level, the supports are 15 ft. apart. On 100-ft. radius curves, the supports are spaced at 10 ft., on 150-ft. curves, at 11 ft., and on 250-ft. curves every-14 ft. Both arc-weld and pin-driven bonds are used, depending on the permanency of the track.

Grade crossings have been avoided where practicable and none occur within 2500 ft. of the shaft. Should trouble occur at the shaft, the dispatcher can route all the traffic over the loaded crossovers to the slope and back over the empty crossover, without disturbing a single individual. At intersections in the mine where such construction is impractical, automatic signals are used. These give ample warning by lights in each entry approaching such intersections of the presence of a trip of cars in the other. These lights, of green, orange and red, form a system that is comparable to any used on our major railroads.

When the loaded trips reach the shaft bottom, the locomotives cut off and go through chutes to the right or left. The shunting locomotive, by means of a sliding extension arm engages the cars between the bumpers and moves the trip to the car haul at the dumps. The car haul, or feeder, is a specially constructed chain that engages cast-iron lugs on the under side of the cars. The chain grips the lug from both sides so that the trip can be moved in either direction at will. A 150 hp. motor supplies the power. The dumps are turned by ropes actuated by electric motors. Both the feeder and the dumps are operated by one man, who controls the former with his left hand and the dumps with his right.

As the cars are dumped a fine spray of water is played on the coal; this settles the dust most effectively. This fact is shown by the white walls of the shaft bottom, which have not been whitewashed since coal was first run up the shaft a little more than a year ago. About 2.3 gal. of water per ton of coal is required.

The coal drops into a concrete bin, with a gate below at the opening to the chute that leads to the skip landing; this chute has a gate at the lower end and is designed to hold a skip load. The skip actuates electric contacts in the shaft that start and stop the motors operating the upper or bin gates. The chute gates are mechanically operated by the skip, as it goes into position or leaves the landing. Naturally, the loading arrangement is so designed that successive gates are not open at the same time. A sump elevator has been installed at the foot of the shaft; and this permits easy cleaning of the sump, weekly.

After dumping, the cars run by gravity to the empty track, which is at right angles to the main bottom; there the "empty" shunt locomotive moves them into position for the mine haulage units. No cars are uncoupled except when it is necessary to get a car of slate out of the trip before it reaches the dump.

Timbering

As roof supports, 5-in. H beams are used; the 12-ft. lengths are not too heavy to handle and furnish ideal support. In Fig. 5 is shown the plan of steel timbering at road intersections. With from 1 in. to 10 ft. of draw slate over the coal, the roof is very treacherous and a constant source of danger. Therefore, nothing is overlooked that will tend toward safe

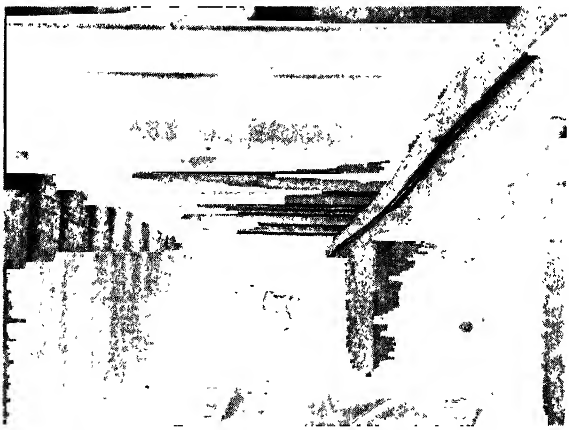


Fig. 5.—Steel timbering at intersecting roads of Nemacolin Mine.

mining. The first rule taught the loader is that after seeing that his place is safe, he must erect a center post to protect him while he is loading out his "cut." Considerable areas of exposed slate along haulage ways have been gunited; undoubtedly, this has avoided the loading out of many cars of slate.

Safety

We believe that frequent inspection by proper officials together with strict discipline will prevent most accidents. All working places are inspected twice daily by the fireboss, twice by the assistant mine foreman for that section, and once by the night fireboss before the cutters come in. This is irrespective of the visits of the mine foreman, his general assistant, or the safety inspector; who cannot visit each face every day. When the men learn that they will be strictly disciplined for infractions of rules, they will be ready at all times to cooperate with the management to the highest degree of industrial safety and efficiency.

A central safety committee, consisting of department heads, meets with the superintendent once each month; they discuss recent accidents and any feasible means for preventing them in the future. In addition, possible areas of danger around and about the plant are discussed and plans made for their elimination. After this meeting the mine foreman assembles the underground organization, tells them the subjects discussed in the meeting of the central committee and asks for suggestions. The thirty or more men in this group are encouraged to talk freely and many valuable ideas are brought out. The plant safety inspector together with the mine foreman attends both meetings and perfect coördination is maintained.

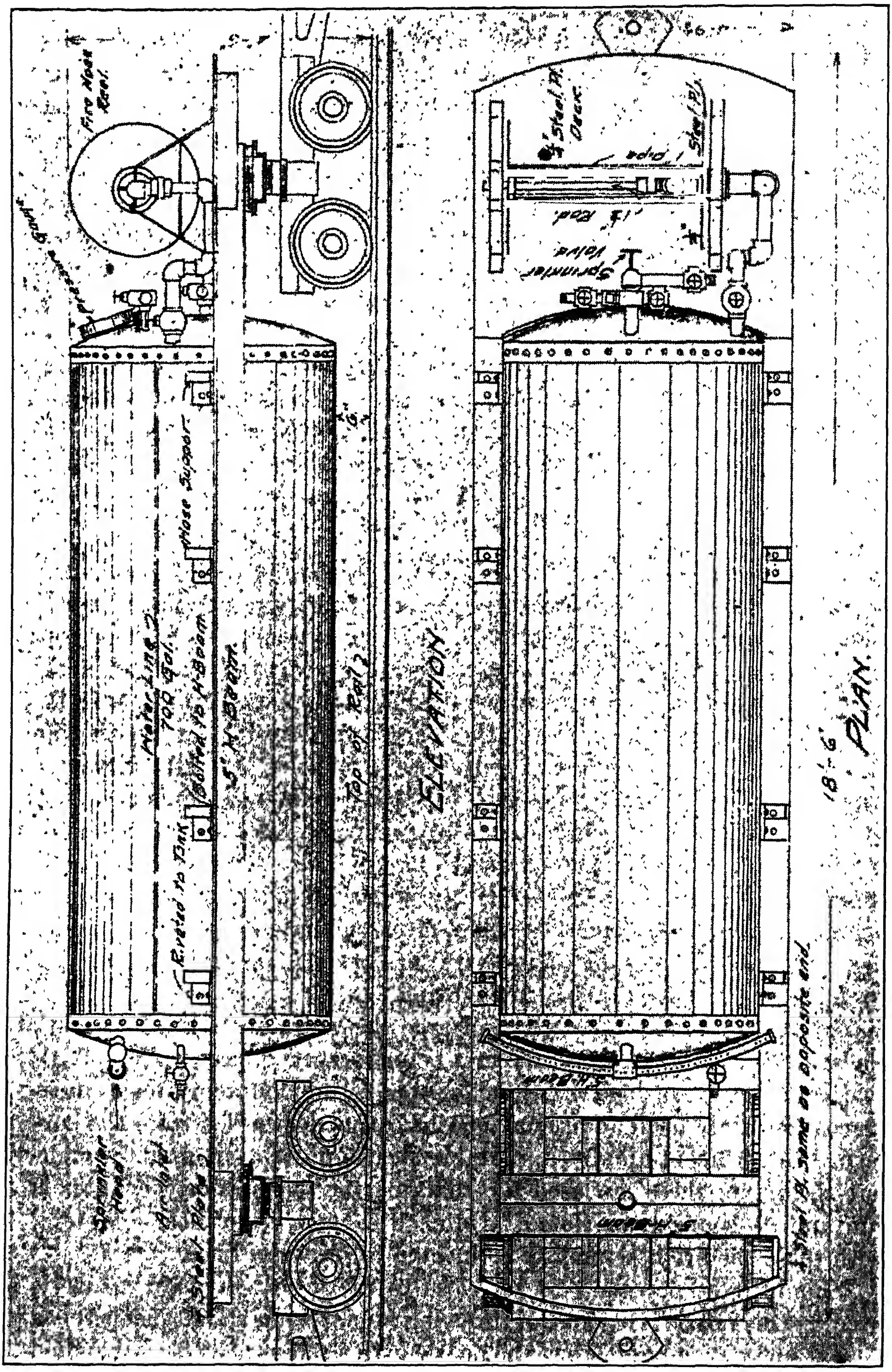
In addition to the safety inspector's report on accidents, an underground committee also makes an investigation and report; this committee consists of an assistant foreman, a transportation man, and a loader. The personnel of this group is changed periodically by the mine foreman.

Fire Protection

The first fire-fighting equipment, on a large scale, was the tank car; two of these were built several years ago and have been extensively used. This car consists of a cylindrical tank, with bowed ends, 11 ft. long and 4 ft. in diameter, that is suspended horizontally between 5 in. H-beams, which are mounted on two platform trucks. Its overall length is 18 ft. 6 in., the height is 4 ft. 6 in., and the truck width is 4 ft. 9 in.; yet it will pass over a 16-ft. radius curve with ease. It holds 700 gal. of water, which fills it slightly over two-thirds full; the remaining space is occupied by air under 100 lb. pressure, Fig. 6. At one end, mounted on the truck is a reel holding 200 ft. of standard 2½ in. fire hose. This reel has a tank connection along its axis and is directly connected to the hose, so that the operator needs but to open his valve, after pulling out the hose, and he has a stream of water under a pressure that bears a favorable comparison to that present in most surface fire fighting.

As the tank will completely discharge the water in from 3 to 5 min., the second tank is necessary, the first being recharged meanwhile. With two cars, 400 ft. of hose is available; this will undoubtedly reach most places, certainly until additional lengths can be forwarded from the surface. Government-approved, portable air compressors are used throughout the mine; these are used for charging the tanks after the water is run and its connections closed. On each car is a set of hose adapters; with these the fire hose may be connected to the water lines.

Should it be necessary to get close to the fire a Hayward nozzle is used. This nozzle is capable of throwing four streams simultaneously—a lateral spray at right angles to the nozzle with a radius of 35 ft.; a diagonal lateral spray just ahead of the first named, another diagonal spray at the end, and a direct stream. The nozzle has several detachable rings which


Fire Hose Reel
Pressure Gauge
Water Line
700 Gal.
Hose Support
Riveted to Tank
Bolted to H-Beam
5" H-Beam
Top of Rail
Sprinkler Head
Air Inlet
Steel Plate
ELEVATION
Sprinkler Valve
Steel Pl.
18'-6"
PLAN.
Steel Pl. same on opposite end

vary the flow so that twelve combinations of streams may be obtained. This nozzle is kept in a glass box in the lamp house so that it can be seen and its availability is known to all employees. Its great advantage is that three sheets of water will drive smoke ahead of the hoseman, enabling him to reach the scene of the fire with comparative ease.

Each working face contains a 1-in. water line and a 2-in. line is close behind so that water for refilling the tanks is always available. This sprinkling system is under a constant pressure from a tank on the surface and is not less than 100 lb. at any time. Each face also has a sprinkling hose (½ in.), so that in most cases a fire can be extinguished without calling on the tanks. The main sprinkling line is a 6-in. pipe subdividing into 4-in. and 2-in.

In addition each of the thirty-four locomotives is equipped with a carbon tetrachloride extinguisher and in each section of the mine, several

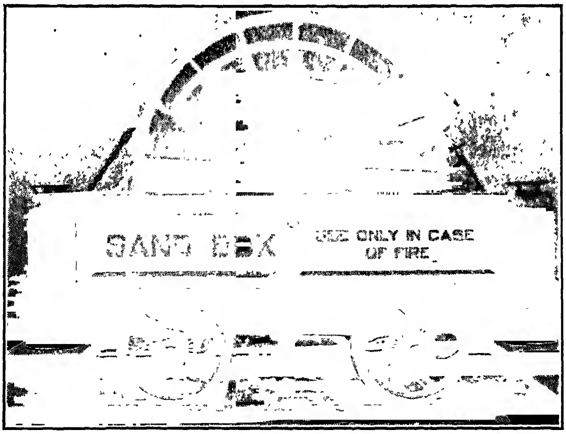


Fig. 7.—Flat car equipped with fire extinguishers and sand box.

2½-gal. carbonated solution sulfuric acid extinguishers are placed. This latter equipment has prevented several serious fires. Defective blasting caps igniting the explosive and thence the coal; a roof fall on a battery locomotive short-circuiting the cells and another on a trolley wire short-circuiting on the rail and igniting the loose roof coal lying about, are examples of potential mine fires that these 2½-gal. extinguishers have smothered. These extinguishers are distributed in a ratio of one to every 100 tons of production. Lastly there are Foamite 40-gal. tanks. These are necessary adjuncts to the tank cars as fires can easily occur in places where no track is laid, such as air courses, and which might assume tremendous proportions were it necessary to wait until sufficient hose was rushed in from the surface. These tanks are mounted on wooden carriage wheels and are whole placed on small flat cars so that they can

be taken by rail as close as possible to the fire. They can be pulled off the truck and taken about like any two-wheeled cart. On each flat car, in addition to the Foamite engine, are four 2½-gal. extinguishers, a box of sand (Fig. 7) and a box containing shovels, picks, axes, saws, hatchets, bars, nails, and a sledge hammer. These boxes are kept locked and sealed, but are easily broken open when necessary.

There are also over 30 automatic reclosing section circuit breakers, together with several hundred sectional line switches, while these are helps in handling fires they cannot be classed as fire-fighting equipment.

On the surface, numerous fire hydrants and hose houses are scattered throughout the town. Each hose house has 200 ft., or more, rubber-lined hose and is inspected weekly. The fire hydrants are oiled and flushed once in six months. A motor fire truck is given weekly inspection. Push buttons in glass-enclosed boxes are placed on electric light poles. These are connected to a large siren that is used for fire alarms only. This can also be operated from the mine office, should any one send in an alarm by telephone rather than from a signal box.

Lamp House and Tipple

The lamp house is considerably larger than is found at most mines, Fig. 8. It is a double one; that is, lamps are distributed on either side so that 1000 men can be handled in 30 min. The men enter, receive their lamps, punch the time clock and proceed to the shaft without back tracking or confusion.

The shaft elevator system is operated like those in any office building, the power plant being in the hoist house. This elevator will handle men as fast as the lamp house and it eliminates a top and a bottom man as well. It is equipped with a telephone and electric lights.

The lamp house basement is an open, well-lighted floor divided into two large rooms. One is the "first aid" room and has a section for equipment. The latter includes ten sets of mine rescue apparatus, all mounted and ready for instant use, together with an oxygen pump, additional tanks, life line, portable telephones, gas-testing equipment, etc. The main portion of the room is used for "first aid" lectures. The second room is an air-tight compartment in which the various rescue crews have their monthly training. It is equipped with an exhaust fan for removing the noxious air after a training period. Formaldehyde is usually burned to create "atmosphere."

There are two tipples. One is at the slope and used for rock and domestic coal or any other coal that is to be segregated; the other is at the skip shaft and handles the great bulk of the tonnage. The slope for several years was the main opening. The haulage rope comes up the slope, passes around a vertical sheave and back to the engine house which is above the tipple, on stilts as it were. This construction was made

necessary by the topography and the proximity of the river. The entire structure is of steel and concrete; the surmounting engine house is of brick. A bin is under each rotary dump. These dumps operate singly as one is for slate and the other for coal.

The shaft tipple is far more pretentious and is built of steel with corrugated asbestos sides and roof. A concrete shroud, separating the skip ways from the tipple, is continued from the shaft head to the dump 110 ft. above. The skips dump into a receiving bin, under which are two pan feeding conveyors. These convey the coal to the main chutes, the floor of which consists of screen bars 1½ in. apart. The smaller sizes drop through while all above is classed as lump and passes over the lump picking

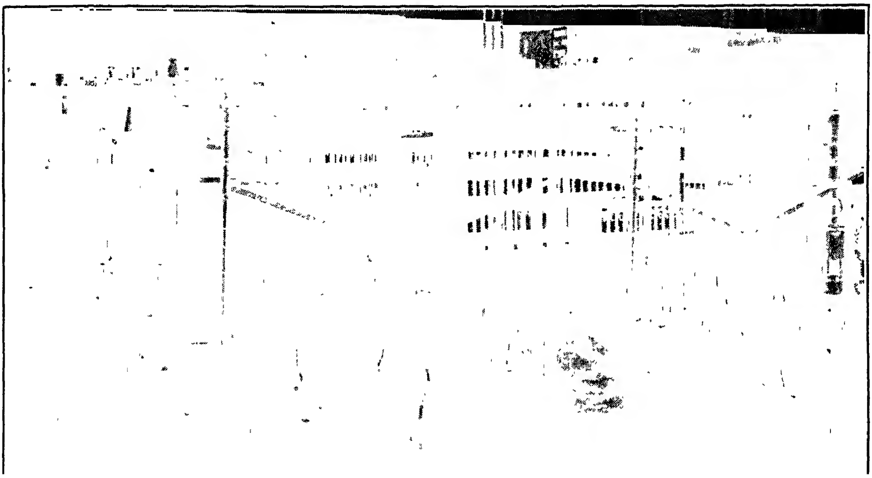


Fig. 8.—Interior view of lamp house.

tables. The undersize flows to eccentric shaking screens, which separate it into two sizes, egg and slack. These are separately conveyed through the picking room but neither receives further cleaning at this time. Six conveyors—two lump, two egg, and two slack—feed two mixing conveyors running at right angles to the former. These convey the coal to the loading chute. All motors in the tipple are controlled from one horizontal panel or switchboard in the picking room. Here the tipple foreman can observe all movements and stop operations or signal the hoisting engineer in a moment. He is also directly connected by telephone with the dumper at the shaft bottom.

At the railroad car loading chute there is an arrangement similar to that at the shaft bottom. The car haul mechanism is under the surface between the tracks and is driven by a 150-h.p. motor. As the yard tracks are level, the operator has perfect control and can move the cars either way. After loading, the cars drop, by gravity, over the scales, where

they are automatically weighed. They then are picked up by a yard locomotive and placed in the lower yard for shipment to Youngstown. All equipment is inspected weekly by a plant machinery inspector.

Slate

The slate is taken from the slope tipple in a 21-ton larry, that operates by third-rail contact while in the plant yard, then by overhead trolley to the dump a mile away. This larry is of turntable construction and, together with the plated belt, can discharge the refuse 35 ft. from the center of the track.

Plant Buildings

The machine shop is modern and properly equipped with lathes, traveling crane, drill presses, milling machine, wheel press, shaper, power hack saws, forge blowers, water stills, air hammer, etc. The storage-battery locomotives are brought out by way of the slope and charged at the shop each night. A mine-car repair shop adjoins the main building.

The emergency hospital is a well-lighted frame building with an operating room, sterilizing room, a four-bed ward, nurses' rooms, bathrooms, etc. Two physicians, a day and a night nurse are in attendance. Each employee, immediately upon receiving an employment slip, must undergo a physical examination and be passed by the doctors before the slip is honored by the payroll clerk. This examination is conducted in a hospital room, especially designed for the purpose.

A club house with accommodations for 13 is close to the plant office and is properly equipped and maintained.

A brick road was built, at a cost of $50,000, to connect the plant with the town which is on the hill above. The houses are probably a step ahead of those at most mines and the layout was classed by the U. S. Coal Commission, in a review of the modern mining towns, as No. 1.

The Mt. Union Sand-flotation Plant for Preparing Bituminous Coal

By T. M. Chance, Philadelphia, Pa.

(New York Meeting, February, 1926)

The first bituminous coal cleaning-plant to use the sand-flotation process[1] was placed in operation on Oct. 1, 1925, at the tipple of the East Broad Top Railroad & Coal Co., at Mt. Union, Pa.

The general principles governing the design of plants of this type and typical operating results of anthracite sand-flotation units have been previously outlined.[2] I will, therefore, confine myself to a description of this particular plant and to some of the elements of design it incorporates.

The Original Mt. Union Tipple

The plant was installed to transfer coal from steel narrow-gage cars of 60,000 and 70,000 lb. capacity to standard-gage equipment. The coal was hauled about 30 miles from the East Broad Top field.

This coal, mined by the Rockhill Coal & Iron Co. and other operators, is a low-volatile slow-coking coal with ash of high-fusion point. It is normally a high-grade coal that does not contain an excessive amount of free impurities, but it requires careful handling to secure a fair domestic yield, and, naturally, the need for narrow-gage transshipment increases the difficulty of the problem.

The narrow-gage cars were discharged into an 8-car track hopper equipped with two feeders passing the coal to two 42-in. apron intake conveyors. Twin units were installed throughout the original plant, two lump-and-egg shaking screens, two lump picking-tables, two egg picking-tables and two stoker conveyors. The stoker coal was loaded without cleaning. The loading tracks and booms were in duplicate and, with the layout as originally installed, either unit could be run independently, each unit shipping a different class of coal or both units making the same product.

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[1] T. M. Chance: A New Method of Separating Materials of Different Specific Gravities. *Trans.* (1918) **59**, 263.

[2] T. M. Chance: Application of Sand-flotation Process to the Preparation of Bituminous Coal. *Trans.* (1924) **70**, 740.

Later, a fifth loading track was added to the siding. The tipple was then arranged to make the following products, or combination of products, on five loading tracks:

Run of Mine	
+10 in. lump	Hand picked
+4 ½ −10 in. lump	Hand picked
+2 − 4½ in. furnace	Hand picked
+1 − 2 in. range	Hand picked
0 − 1 in. stoker	Not cleaned

This remodeled tipple had a maximum capacity of about 300 tons per hour. The tonnage was limited by the ability of the picking-table labor to clean the coal.

Flow Sheet and Design of Present Sand-flotation Plant

The difficulty either of increasing the hourly output of the tipple or of insuring uniformity of the product, if the hand-picking units were further extended, led to the adoption of the sand-flotation process for the coal that passed through a 4½-in. round perforation and was larger than ⅜ in. The lump picking-tables were kept and the fine coal was allowed to bypass in the raw state.

This procedure is in line with the principle previously[3] advocated for all coal amenable to such treatment. It eliminates the moisture difficulties due to sludge in wet washeries and to dust in dry processes. The lump can readily be marketed and is of sufficient purity to permit economic hand picking. The coal through 4½ in. can be efficiently washed by sand flotation without crushing to free impurities. Even if this were not so, we would still prefer to wash the coal in as large a size as possible, and to crush the refuse and return it to the original feed, or to other units, to recover the coal.

The tipple was, therefore, redesigned to function as follows. Fig. 1 shows the general layout and Figs. 2 and 3 the flow sheet.

Lump Coal and Feeding

The design adopted called for a total shipping capacity of 500 tons per hour. The raw coal is brought in by the two original 42-in. apron conveyors and is passed to a single lump coal shaker 6 ft. 0 in. wide, making 4½-in. round hole undersize, +4½ by −10-in. lump and +10-in. oversize lump. Both sizes of lump pass to individual picking-table loading booms and are loaded out on separate tracks; the +4½- by −10-in. lump bars are covered when +4½-in. mixed lump is shipped and all lump is then picked and shipped on the one boom. Suitable gates are pro-

[3] T. M. Chance. *Op. cit.*

vided in the conveyor discharge chutes to permit loading of unprepared run-of-mine coal.

Cleaning and Screening 4½-in. Undersize

The 4½-in. undersize at maximum capacity exceeds 400 tons per hour. It is elevated by a scraper conveyor to a battery of four vibrating

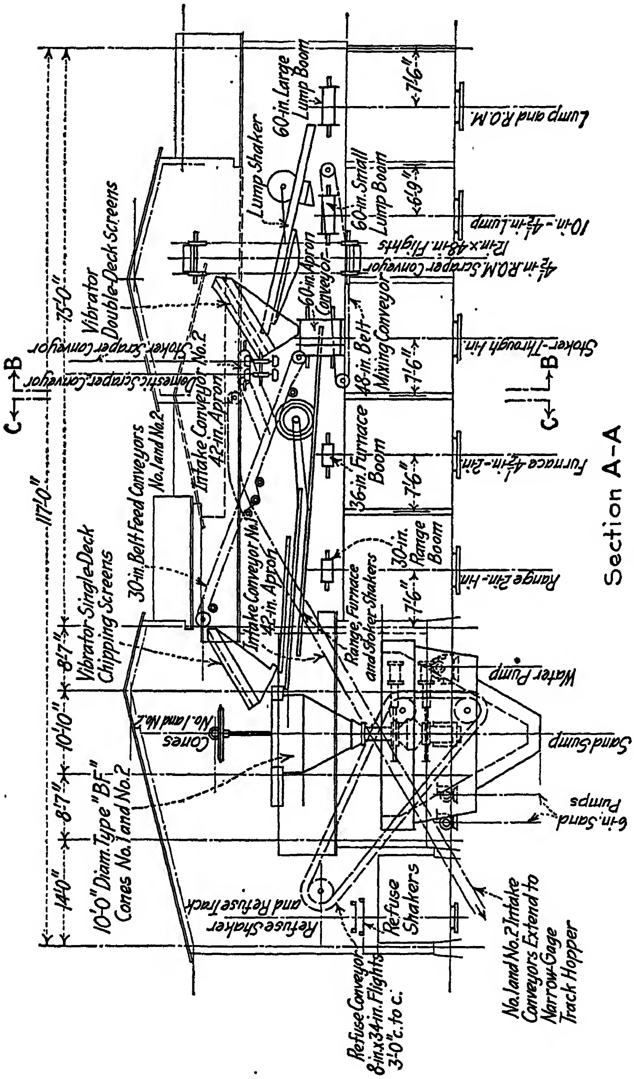

Fig. 1.—Mt. Union Plant of East Broad Top R. R. & Coal Co. for cleaning bituminous by sand-flotation process. (Section A-A.)

screens. These screens, developed by Elmer Shiley, foreman of the plant, consist of two tandem double-deck screen frames placed at a suitable angle (±30°), and vertically vibrated at high speed by eccen-

trics of small throw. The shafts driving the two frames are driven in proper time by a connecting belt. The frames are dressed with wire-mesh cloth to produce plus 1-in. material on the upper deck and plus ⅜-in. on the lower deck. They thus serve to make the preliminary

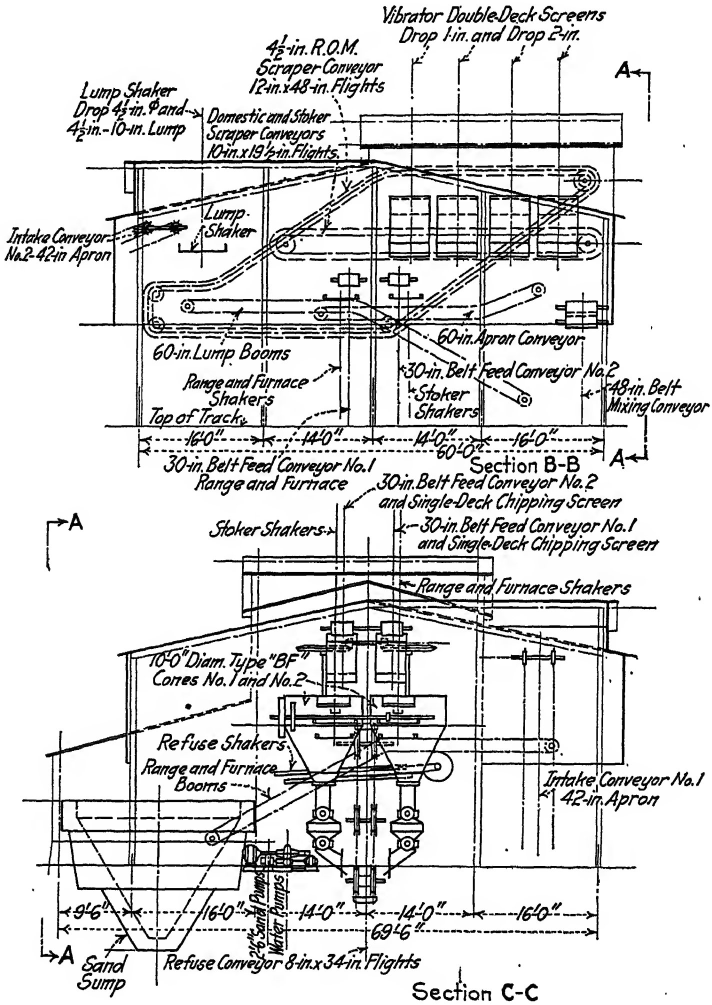

Fig. 2.—Mt. Union Plant of East Broad Top R. R. & Coal Co. for cleaning bituminous by sand-flotation process. (Sections B-B and C-C.)

separation into domestic and stoker size and at the same time to remove the stoker undersize that is to be bypassed unwashed.

The domestic and stoker coal produced as oversize from these screens are gathered by scraper conveyors and delivered to the 30-in. belt of the

domestic and stoker feed-conveyors. These two belt conveyors transport and elevate the two grades of material to the sand-flotation section of the plant. They each discharge over single-deck vibratory screens, of the

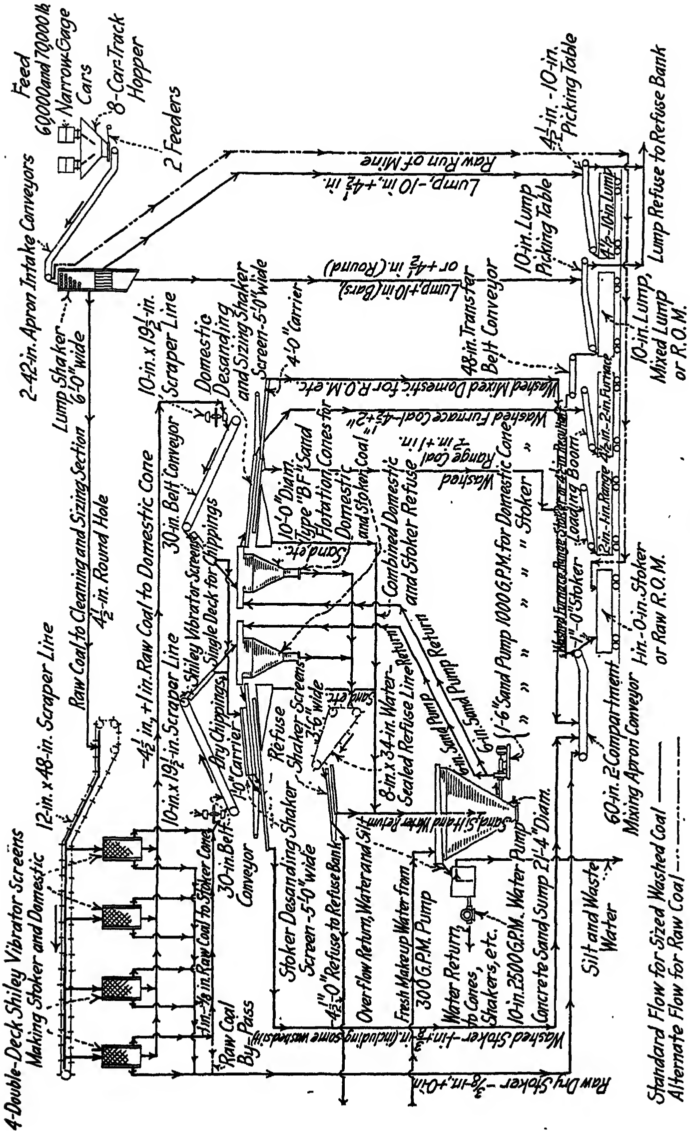

FIG. 3.—FLOW SHEET OF MT. UNION TIPPLE.

character described, to the sand-flotation cones. A small amount of undersize, produced by degradation is removed by these screens and sent to the washed stoker coal.

Separating Cones

One separating cone is used to wash domestic coal and another for stoker coal. This permits the use of different densities for the different grades of coal. The cones are 10 ft. in diameter and in general construction are similar to the units used to wash anthracite coal. A detailed description of the special features of units used for bituminous preparation is given later in this paper.

The cones are provided with 6-in. sand pumps that return the sand and water mixture used for the overflow of the washed coal, which overflow passes to shaker screens provided for each cone. The domestic screen desands the coal and separates it into $-4\frac{1}{2}+2$-in. furnace and $-2+1$-in. range; the range deck is fitted with desanding segments. The furnace and range coal are discharged from this shaker to loading booms serving the furnace and range tracks. The range deck of the domestic coal shaker is also extended as a shaking chute to convey furnace and range to a 60-in., double-service apron conveyor, for the shipment of run of mines and mixed lump.

The stoker screen serves to desand and dewater the washed stoker coal and convey it to the 60-in. apron conveyor. The latter conveyor is placed below the undersize discharge from the four vibrating screens and the ⅜-in. raw (dry) coal passing as undersize from the lower deck of these screens is collected in a separate compartment thereof, into which compartment the washed stoker is also delivered. There is, therefore, a premixing of the wet and dry stoker during loading from the conveyor into the railroad car before the final mixing occurs. The conveyor is provided with two compartments so that one may be used solely for furnace and range coal when they are shipped mixed with lump coal and without admixture of stoker coal.

When either such mixed lump or run of mine is to be loaded, the 60-in. apron conveyor carries the washed coal to a short 48-in. belt conveyor which transfers the mixed washed coal across the tipple to the lump-coal loading track and discharges at the end of the lump boom.

Refuse Handling

The refuse discharged from the two separating cones passes to a common water-sealed scraper line which elevates it to the refuse desanding shaker screen. This screen discharges either directly to a narrow-gage railroad car, or to a scraper conveyor, which extends over the center line of the refuse loading track and thus permits refuse empty-car storage. The refuse passes directly to the bank and no crushing or retreatment is necessary, as not enough laminated coal is present to justify such procedure.

The slate valves, used to trap the refuse from the cones, are operated hydraulically. The operating levers for both cones are so placed that one slate-valve runner can attend both cones.

Sand Return and Water Circulation

The sand and water reclaimed by the domestic stoker and refuse shakers are returned by wood-lined chutes to a common sand sump. This sump is built of concrete, 21 ft. 4 in. inside diameter, with the top of the water-leveling strip at an elevation of 9 ft. 2 in. above datum or top of rail of the loading tracks. The tops of the cones are at an elevation of 23 ft. 0 in., so that the vertical head through which the sand must be raised in returning it to the cone is but 13 ft. 10 in. For this return, a 6-in. centrifugal sand pump is installed with each cone. These pumps are directly connected to 25-hp. motors and have a capacity about 1000 gal. per minute.

Makeup sand is supplied to the sand sump to maintain the required volume of sand in circulation. A concrete sand-storage floor is provided adjacent to the sump and served by the range coal loading-track. This provides storage for several cars of sand. The small quantity required each day is delivered to the sand sump by hand.

Water Circulation and Sludge Elimination

Clear water, in addition to the sand-pump return, is required for agitating the fluid mass, for shaker sprays and for clearing pipes and valves of sand. This water, with the return from the two 6-in. sand pumps, is delivered into the sand sump through a common central vertical conduit terminating well below the top of the sand sump. The sand and the water necessary to pump it pass down to the suction pipes of the sand pumps and the clear water, carrying the greater part of the fine coal sludge that enters the system, rises and flows into a rim overflow launder. This launder discharges into a small concrete suction sump serving a 2500 g. p. m. single-stage centrifugal pump, direct-connected to a 50-hp. motor.

The suction sump is not large enough to function as a settler, but is nevertheless fitted with an overflow through which the excess makeup water added to the circulation is discharged. This overflow is in the form of a conduit that removes the excess water from the lower part of the sump and prevents sedimentation of the sludge. The suction sump is thus intended continuously to reject an overflow of average sludge content similar to that circulated by the main water pump.

Sludge can also be drawn off from the upper part of the sand sump, as this sump classifies much of the sludge into the upper part.

This simple method of sludge elimination is possible because the fine coal that enters the fluid mass of sand is chiefly the dust adhering to the coal in the feed. Minimum degradation of the coal occurs in the washing process, hence a minimum of sludge is produced internally.

The makeup water supplied to the plant is pumped from a nearby creek with a 300 g. p. m. pump. This makeup has been successfully reduced to less than 200 gal. Part of this water is carried away by the washed coal, refuse and leakage, and the remainder overflows to waste from the suction sump. Hence, if recovery of this waste water and sludge were desirable, a very small settler would be used and the small volume of wet sludge recovered could be mixed with the washed coal.

A considerable part of the fine coal, continuously circulated by the water pump and pumped through the coal-shaker spray pipes, intermixes with the washed coal on the screens and is thus removed from the circulating system. The rear part of the desanding screen carries very little coal and this screen continuously acts to remove, as oversize, fine coal that would be driven through the screen if much coarser coal were present. The action is the same as that previously described[4] with reference to desilting sand in anthracite cleaning, and occurs continuously during plant operation.

Desanding Shakers

The clean-coal shakers for the domestic and stoker cones are duplicate in general construction. They each consist of a top deck, 5 ft. 0 in. by 25 ft. 10½ in., and a bottom deck 5 ft. 0 in. by 23 ft. 0 in., with a 4 ft. 0 in. by 17 ft. 3 in., shaking-chute extension, the bottom deck being, therefore, 40 ft. 3 in. long. The domestic shaker is fitted with a return chute that carries the washed range back to the range boom. The stoker shaker is provided with a 12-in. steel shaking-chute that conveys the dry undersize, from the domestic and stoker vibrator feed-screens, above the sprays so that it finally mixes with the washed stoker on the shaking-chute extension. A 5 ft. 9 in. desanding section is on the upper deck of each shaker, and a 17 ft. 3 in. section on the lower deck, all desanding segments being bronze plates with $\frac{3}{32}$-in. holes. Suitable fly gates are provided to make the various combinations of sizes that may be desired.

These two sets of shakers are balanced in a way that we have adopted at some recent anthracite plants. The lower deck of each shaker is driven by eccentrics at 180° to those driving the upper deck, which tends internally to balance the screen in the usual way. The two screens are driven from a common eccentric shaft with the lower decks at 180° to

[4] T. M. Chance. *Op. cit.*

each other and the upper decks likewise at 180°. Therefore, each pair of lower decks and each pair of upper decks act as couples in a horizontal plane, the upper deck couple being of opposite sign to the lower deck couple. The only tendency to twist in a horizontal plane is due to the difference in weight of the top and bottom decks, which the gyroscopic action of a 48-in. flywheel on the shaft partly compensates. This "couple" balance with the internal balance of each screen results in very smooth running and few delays from broken arms or hangers.

The refuse shakers are smaller than those that we have used at anthracite installations; they consist of a 3 ft. 6 in. by 11 ft. 6 in. top deck dressed with $\frac{3}{32}$-in. segments, and a 4 ft. 0 in. by 11 ft. 6 in. blank bottom deck carrying sand and water to the sand sump.

All the shakers have wooden sides and are hung on 1 by 8-in. oak boards. They are driven at 160 r. p. m. by wooden arms with flexible wooden spring pieces, which are attached to the shaker by a 3 by 3 by $\frac{3}{8}$-in. steel angle 1 ft. 6 in. long. This new type of arm, recently designed by Paul Sterling, mechanical engineer of the Lehigh Valley Coal Co., has given excellent results.

The smooth operation of these shaker screens has been due largely to the care and skill in erection of the local colliery personnel. Screens of this type cannot be successfully operated unless they are properly erected, as high speed is impossible to maintain when arms and hangers are in poor alignment.

Power

The total installed motor capacity (Table 1) is 468 hp., of which 240 hp. serves the washery and screening section, including the power necessary for the 30-in. feed conveyors and makeup water pump. The remaining motor capacity is required for the raw-coal feed conveyors, lump shakers, vibratory screens, loading booms and transfer conveyors.

The cleaning section is driven by one 75-hp. squirrel-cage motor. This motor, through a main-jack shaft, drives 2-cone agitator shafts, both 30-in. belt feed conveyors, the refuse conveyor and the refuse shaker. Friction clutches are installed on the necessary individual drives. The domestic and stoker shakers are driven by one F. T. R. 30-hp. squirrel-cage motor of special design that secures high starting torque without excessive starting current and eliminates the usual starting devices.

The six vibratory screens are driven by six individual 3-hp. squirrel-cage motors, mounted on the screen frames and connected by V belts to the screen eccentric shafts.

TABLE 1.—*The Motor Distribution at the Mt. Union Plant*

Use	Type	Drive	Hp.
No. 1 feeder	Squirrel cage	Unit belt	15
No. 2 feeder	Squirrel cage	Unit belt	15
No. 1 apron conveyor	Squirrel cage	Unit belt	30
No. 2 apron conveyor	Slip ring	Unit worm	30
Lump shaker	Squirrel cage	Unit belt	20
All transfer scraper and apron conveyors	Squirrel cage	Group chain	40
48-in. belt conveyor	Squirrel cage	Unit worm	5
6 vibratory screens	Squirrel cage	6 — 3 hp. V belt	18
Picking-table and loading booms	Squirrel cage	Group gear	35
Boom hoists	Squirrel cage	Unit hoist	20
Total feeding, primary screening, conveying and loading			228 hp.
Cones, 30-in. belt			
Conveyors, refuse			
Shakers, and conveyor	Squirrel cage	Group belt	75
Clean-coal shakers	F. T. R.	Unit belt	30
No. 1 sand pump	Squirrel cage	Direct connected	25
No. 2 sand pump	Squirrel cage	Direct connected	25
10-in. circulating pump	Squirrel cage	Direct connected	50
Slate valve pump	Squirrel cage	Direct connected	20
Makeup	Squirrel cage	Direct connected	15
Total cleaning and sizing			240 hp.
Total connected load for entire tipple			468 hp.

When the plant is shipping 500 tons per hour, the connected load per ton-hour is, therefore, 0.936 hp. Of this tonnage about 330 passes to the cleaning sections for which 240 hp. are installed, so that the connected load per ton-hour of washing and sizing capacity is about 0.728 hp.

LABOR

Final economies in force account will not be attained until all the operating units have been properly coördinated, such as centralized motor control room and car-handling devices. However, the present operating crew is about 15 men less than that formerly required for a much smaller output of cleaned coal.

PRESENT AVERAGE CREW

	Number of Men
Weighman, narrow-gage unloading, broad-gage loading and car-patching	7
Lump pickers	4
Booms and tipplemen	2
Machinery attendants and repair crew	3
Electrician	1
Vibratory screens	1
Cone runner	1
Slate valves	1
Pump man	1
Refuse disposal	1
Inspection	1
Foreman	1
	24

Accurate comparison of present and former force account is difficult because the size of the old crew naturally depended on the number of pickers necessary and also on the overtime worked. The present crew can ship 500 tons per hour of prepared coal continuously, whereas the former crew could not prepare over 300 tons per hour and, even when so doing, could make but little improvement on the range coal and none on the stoker, which was a very large part of the total output. The man-hours per ton of *clean* coal, under present conditions with every size of coal prepared to a predetermined ash standard, therefore, is much lower than would appear from a simple comparison of the relative force accounts.

TYPE OF BUILDING

The original tipple was of steel construction with corrugated black-iron sheathing. The additions to that structure and the new sand-flotation section were designed in accordance with the standards which we have adopted in our anthracite plants. All columns are rolled **H** sections, and beams subjected to horizontal stress are either Bethlehem girders or **H** sections. Horizontal vibrations are suppressed by introducing trusses in the floor planes.

This type of construction, with the low height of building (41 ft. 1¼ in. to ridge beam) and an extended use of panel bracing, has resulted in freedom from objectionable vibration, notwithstanding the considerable length of the clean-coal shakers.

DESCRIPTION OF SEPARATOR CONES

The cones used for the domestic and stoker coal are duplicates and follow the general design of the units used for the preparation of anthracite. The former, however, have a much smaller ratio of throat diameter to top diameter, as these 10 ft. 0 in. cones are provided with the

22-in. diameter discharge throat used on the type-F, 13 ft. 6-in. diameter anthracite cone.

The largest permissible diameter of throat and classifier column is obviously desirable because the maximum size of coal that can be washed depends on the maximum diameter of refuse particle that can safely be discharged through the throat and classifier column. The limiting factor on the throat diameter for any given top diameter is the volume of agitation water needed to obtain the desired density. This limit exists because the water passing up through the classifier column on its way to agitate the fluid mass prevents the unrestrained fall of the sand through the classifier column; hence a certain diameter cannot be exceeded for any given volume of water passed per minute up the classifier column without excessive dropping of sand with the refuse. The volume of such agitation water, moreover, is determined by the area of the top of the separating cone for any given specific gravity, because this gravity is produced by a fixed quantity of water per unit of cone area.

With the high densities used for anthracite which permit relatively little water per square foot of area, a much smaller discharge throat and classifier column are necessary for any given top diameter of the separating cone than with the lower densities used for bituminous coal. Bituminous cones of small top diameter, therefore, can be used with comparatively large discharge throats and refuse valves, which is of peculiar advantage in bituminous washing. To wash large pieces of coal may often be desirable and these basic requirements of design automatically meet such washing requirements.

The comparatively large volume of water necessary for the low densities required in bituminous washing may often produce upward velocities in the classifier column that would not only prevent the sand from falling, but would also restrain the smaller slate unless a column of excessive diameter were used. This condition is overcome by supplying the water for agitation both through the classifier column and at a number of points in the sides of the cone connected to suitable manifolds. With this arrangement we have been able to produce a wide range in density without interfering with the operation of the classifier column. Further, this arrangement has the advantage of eliminating the introduction of an excessive quantity of water at any one point in the fluid mass. Such excess water would reduce the distributing effect that must otherwise be produced by the rotating agitator arms.

Agitators

In the cones used at Mt. Union, three of these agitator manifolds are incorporated in the body of the cone. This construction is cheaper than that of separate pipe manifolds with individual connections, and the erection labor is reduced. The manifold channels are formed by an

angle bent to the curvature of the cone shell and welded in place. A triangular waterway is thus provided with but one field connection per manifold.

The mechanical agitators are driven by bevel gears and are of lighter construction than those used in anthracite machines. The agitator arms consist of two groups of four steel angles attached to spiders keyed to the 4 7/16-in. agitator shaft. The bevel pinions are driven by friction clutches through chain drives from a common jack shaft. The average operating stresses in bituminous cones are lower than in the anthracite machines because of the great fluidity of the low-density fluid mass used.

The slide valves, used for trapping the refuse from the base of the classifier column, are operated by 13.5-in. diameter hydraulic thrust cylinders. Hydraulic water is circulated by a 2-stage centrifugal pump of 120 g. p. m. capacity; sufficient air-chamber capacity is provided to give the necessary quickness of stroke. Standard 4-way operating valves are used and are mechanically interlocked to prevent opening both valves simultaneously.

Operating Results

Tonnage

The Mt. Union plant shipped 62,000 tons of coal during October, the first month of operation. The maximum hourly production during this time was 515 tons in a day during which 3865 tons were shipped in 7½ hours operating time. Daily outputs in excess of 4000 tons have been reached, but such tonnages cannot be maintained until the mine capacity has been increased in accordance with the schedule contemplated in the design of the new equipment.

The plant operated smoothly from the start and required no alterations or modifications in original design, which, of course, is self-evident from the record of monthly tonnage.

Sand Data

Desanding and sludge elimination are done more readily than in many of the anthracite plants using the sand-flotation process, because the desanding process is not at all a true screening operation, but rather consists of straining a fluid mixture of sand and water through holes that are immensely larger than the maximum size of sand grain; most of the sand is washed through the desanding segments by the overflow water and with little aid from the shaker sprays.

The sand used, therefore, must be coarse enough to settle readily so that it can be diluted with large quantities of water for desanding and be quickly reconcentrated for use as separating medium for the fluid mass.

In the anthracite plants washing rice and barley, the fine coal blinds the desanding segments so that a much greater area of desanding screen

is used. In the same way, the anthracite units introduce more fine sludge because of the smaller size contained in the feed. This difficulty is, however, somewhat offset by using on the anthracite feed shakers sprays that thoroughly wash off the dust. I am quite certain that in those future bituminous plants that wash the fines, the desanding and the sludge elimination will still be more readily effected than in anthracite plants treating similar material, because the lighter bituminous coal does not so easily blind the screens and is more readily removed from the sand sump. This difference can be observed in the behavior of the fines that now enter the circuit at Mt. Union.

The sand consumption is very low, less than two cars of sand being used in the first 100,000 tons shipped, or about 1.0 lb. per ton of total shipment, or 1.5 lbs. per ton washed. The daily makeup sand amounts to from 2000 to 4000 lbs., depending on the tonnage shipped, and is too small to require mechanical handling. We believe the small sand consumption is largely due to the low density mixtures circulated and the high cleaning capacity of a comparatively small unit. The sand is of the same specification as that used at the anthracite plants. It is obtained from producers of silica glass sand.

Proportion of Sizes Shipped

Of the total tonnage fed into the plant, over 60 per cent. passes through the 4½-in. perforations on the lump shaker and is retained on the lower deck of the vibratory screens.

From 15 to 20 per cent. of the total feed passes through these screens and is bypassed dry. The remainder is shipped as lump coal or mixed with run of mine. On a basis of 500 tons per hours, the coal passing to the washery is in excess of 300 tons and this is divided between the domestic and stoker cones. The ratio of tonnage handled by these machines depends on the character of the run-of-mine coal. It is difficult to determine to just what the hourly production of these cones may be raised but there is no question that each of these machines has shipped in excess of 150 tons per hour.

Analytical Results

The screening of the coal into the domestic and stoker sizes results in a concentration of the impurities in the domestic coal. The analysis of the raw coal of this size is far higher in ash than any of the other sizes shipped. Therefore, the greatest ash reduction is made by the domestic cone, which consequently discharges most of the refuse.

Many analyses have been made of the washed stoker coal, the −⅜-in. raw stoker coal, the mixed domestic washed and raw coal, and the refuse. The character of the feed naturally varies according to the mines from

which it is shipped, but this variation is hardly reflected in the grade of the washed coal produced. We have not included average analyses of the run-of-mine raw coal or the lump, as the large coal contained in these sizes is not sent to the washing section and, except for run-of-mine coal, is not mixed with the washed coal.

	Ash, Per Cent.	Sulfur, Per Cent.
Washed mixed furnace and range (−4½ in., + 1 in.)	7.5	1.10
Washed stoker (−1 in., + ⅜ in.)	7.1	1.22
Raw stoker (−⅜ in., + 0 in.)	7.9	1.10
Shipped stoker (−1 in., + 0 in.)	7.4	1.20

It will be noted that the washed stoker is of lower average ash content than the domestic coal. Control of this kind can be maintained because of the use of separate cones for the two grades of coal, the stoker cone operating at about 1.45 specific gravity and the domestic cone at 1.55. This method of operation avoids the loss of merchantable domestic laminated coal that would occur if this size were washed at the density used for the stoker coal.

These analyses show the desirability of bypassing the fine coal at this particular plant. This −⅜-in. coal constitutes about one-third of the stoker and the average ash content does not raise the ash of the shipped coal enough to require cleaning of the fines. If a lower ash content were desired, this could be obtained (within the limits of the inherent ash of the pure coal) by lowering the stoker-cone density. If the ash content is to be reduced, a shrinkage in weight must be suffered, whether by rejecting more coarse bone or by washing the fines. Although the latter procedure should theoretically give a higher ash refuse, in practice the sludge (or dust) and other losses, to say nothing of the increased equipment required, will often outweigh such possible chemical advantage.

It is not here implied that this method of operation will prove the better for all coals, but I believe it will in most plants. It would probably be impracticable, even with this coal, if the coal were crushed prior to washing, because the fines then would be greatly increased in volume, and further contaminated with crushed refuse. This again emphasizes the desirability of washing as large material as is possible.

An idea of the washing efficiency of the process can be obtained from the following analyses of samples taken simultaneously from the feed and the discharge of the domestic cone unit. The concentration of high-ash material in this size, before noted, is thus made evident, the raw run-of-mine coal containing from 9.5 to 10.5 per cent. ash.

Mixed Furnace and Range Analyses

	Ash, Per Cent.	Sulfur, Per Cent.	B. t. u.
Raw coal (−4½ in., + 1 in.)	17.75	1.58	13,680
Washed coal (−4½ in., + 1 in.)	7.42	1.10	14,580
Refuse (−4½ in., + 0 in.)	58.60	3.59	5,602

As all the sizes are discharged by a common refuse conveyor, the refuse sample includes that from the stoker cone. The amount of stoker refuse is very small and, therefore, does not materially affect the resulting analyses.

Moisture

The free water running down drainage ditches along the loading tracks and under the loaded cars, so much in evidence at many washing plants, is not present at the Mt. Union plant. The stoker coal, as loaded out, is comparable to that at many mines producing some of the coal from dip workings. The domestic cars contain little free water.

This visual impression is borne out by moisture analyses. Average car samples show but 4 to 5 per cent. moisture, which checks with the unaccounted increase in weight, after deductions for refuse, sludge and coal lost in transit. This increase in weight is charged to added moisture and amounted to less than 0.5 per cent. for the first month of operation. The coal in transit from the mine is often apparently as wet as the washed coal normally shipped, as it is frequently subjected en route to a day of continuous rain, but even under these conditions there seems to be no noticeable increase in washed-coal moisture.

On the other hand, cars of unwashed Mt. Union stoker coal sampled at destination gave an average moisture analysis of 4 per cent., which confirmed the belief that under normal transit conditions the moisture added from atmospheric agencies is as great as that introduced by the use of sand flotation at this plant.

I believe that this condition is due to the excellent drainage provided by the clean-coal shakers; the short time the coal is in process, which prevents it from becoming water soaked; and the presence of a minimum of wet sludge.

Quantity of Refuse Removed

The Mt. Union plant sends from 4 to 5 per cent. of the feed to the refuse bank, indicating an ash reduction on the run of mine of about 2½ per cent., which checks with the general figures available. With the former hand-picking equipment, the loss was but little over 1 per cent. and when it is remembered that most of the refuse is contained in the sizes that were picked, the low efficiency of hand picking is apparent. The small loss of weight due to refuse removal precludes the possibility of any appreciable loss of pure coal with the refuse and confirms the accuracy of the refuse tests.

Loss of Fine Coal

Tests made on the solids in the final waste-water overflow from the plant indicate that the fine coal so lost is a fraction of 1 per cent. Fur-

thermore, this coal averages from 15 to 16 per cent. ash, and from 2.0 to 3.8 per cent. sulfur, so that it is not a highly desirable material to reclaim. But little of this high-ash condition can be charged to sand loss as the overflow contains no sand over 80 mesh. Moreover, less than 700 lb. of minus 80 mesh sand is added to the system per day, so that even if all of such fine sand passed to the overflow, it would not greatly increase the ash of the rejected sludge coal. Furthermore, the sand cannot be held accountable for the large increase in sulfur as it contains no pyrite.

This impure condition of the very fine silt is an exact analogue to that existing in anthracite fines. As the coal and refuse are comminuted, a point is reached where the refuse is more easily crushed than the coal, and at such point the curve of ash content plotted against the size of particle shows a marked change in direction, becoming steeper and steeper as it is extended toward the particle of zero diameter.

I believe that Charles Dorrance, Jr., was one of the first to point out this characteristic of anthracite fines, but I do not think it has been fully appreciated by those interested in cleaning bituminous coal. The high-ash content of many bituminous sludges is often attributed to the presence of emulsified fire clay that may be due to communition of the refuse caused by crushing the larger pieces of coal.

I refer to this at some length because it furnishes another reason why unnecessary crushing of the coal should be avoided. Not only are fines thus produced, which at best are difficult to dewater, but these fines are contaminated by still finer refuse which is most difficult to remove economically.

Uniformity of Feed

In most cleaning processes a fair uniformity of feed is essential to proper operation because the machines are set to discharge their products at a predetermined rate, and if this be radically changed, variations in the cleaned product are unavoidable. In the same way, abrupt changes in the grade of the feed tend to prevent efficient cleaning. For this reason, large feed pockets are often installed to smooth over variations in the rate or quality of feed.

Changes of this kind have little or no effect on efficiency in the operation of the cones in the Mt. Union plant, and no adjustments are made to accommodate the plant to them. Without storage pockets or feed bins, the feed stops and starts as the cars are dumped irrespective of the operation of the cones. This feature is of especial value in bituminous washing as car changing and other interruptions to operation peculiar to soft-coal plants occur without disturbing the operating results.

Costs

The cost of the Mt. Union plant was larger than the cost of an equivalent new installation because of the need for remodeling the

existing tipple while maintaining average daily shipments in excess of 2000 tons. This was done by the local colliery organization; the plant was completely converted with the loss of but two days of mine operation.

The cost of the washery addition, including screening and cleaning facilities, and the necessary building and foundations, was less than $50,000. From the available figures of the cost of alterations and additions to the old tipple, it seems probable that a complete tipple for the same service, if designed as an entirely new unit of 500 tons hourly capacity, could be built for $125,000 to $150,000.

Operating costs naturally depend largely upon the tonnage shipped. With an assumed shipment of 400 tons per hour during a month of twenty-four 8-hour days, and with average tipple labor at 80 cts. per hour, the total cost per ton is about as estimated below:

General Data

Tonnage shipped per month	76,800
Hours worked per month	192
Man-hours @ 24 men	4,608
Man-hours per ton	0.06
Assumed cost of complete tipple	$140,000

Total Cost per Ton Shipped

Labor @ 80 cts. per hour	$0.0480
Power @ 0.9 kw.-hr. @ 1.5 cts	.0135
Sand @ 0.0005 ton @ $2.20	.0011
Maintenance, etc. @ $1,200 per month	.0156
Fixed charges, taxes, etc. @ 12 per cent. on $140,000	.0182
Total cost per ton	$0.0964

Of the total labor cost of $0.048 per hour, 75 per cent., or $0.036, is tipple labor that would be required whether the coal were washed or not, and at least 50 per cent. of the remaining cost would likewise be necessary for a non-cleaning screening tipple. The total operating cost of such a tipple would, therefore, be about $0.0602 per ton. If hand-picking were employed to the extent shown as necessary in the former operation of the Mt. Union tipple, an added labor cost due to not less than 25 men would be incurred, or $0.05 per ton, making the cost $0.1102 per ton, and this increased expense would be required for a tipple shipping uncleaned stoker coal and a domestic coal of inferior quality.

Conclusion

Operation of the Mt. Union tipple has proved as practicable many of the factors that had been thought essential to the development of a simple and efficient bituminous coal-cleaning plant. Advantages that appear to us as of especial value naturally have been stressed in the

preceding pages. It is thought that in many plants they will outweigh the disadvantages of wetting the coal to be cleaned.

Wm. Emery, Jr. and H. H. Morris, of the operating company, designed the screening and transfer layouts of the modified tipple and collaborated in the layout of the cleaning sections. Much of the ease with which the plant was placed in operation was due to the care with which they and the Mt. Union Colliery force looked after every detail of construction.

DISCUSSION

E. S. Tompkins, New York, N. Y.—Does the change in proportion of slate to clean coal affect the density of the solution in the tank?

T. M. Chance.—Changes in the refuse content have had no effect on the performance of the plant. The only thing that affects the density of the fluid mass is the spacing of the sand grains (the amount of sand per cubic inch of water).

E. W. Parker, Philadelphia, Pa.—What was the percentage of coal?

T. M. Chance.—In a recent test, the total float in either size of refuse was less than 2 per cent. of the refuse, and some of this float was naturally produced by chipping off of pure coal from the larger pieces of laminated bone and slate.

C. E. Locke, Cambridge, Mass.—Relative to the statement that the inefficiencies of screening do not affect the results because it makes no difference whether or not fine material goes into the cones: Down to what size will the cones separate?

T. M. Chance.—Finer than the mesh of the sand. The water current that maintains the sand in a mobile condition will lift particles of coal considerably coarser than the sand; therefore before a point is reached at which the coal cannot be floated by the fluid mass, because of the fineness of the coal, this finer coal is removed by hydraulic classification. If it is not discharged from the circuit, it will tend to form a separate coal fluid mass above the sand fluid mass used in the process.

In the coal passing a $\frac{3}{32}$-in. mesh, we, therefore, have a combination of sand floatation product and a hydraulic classification product. This material, when recovered on special desilting screens, is uniformly of lower ash content than the average coarser coal.

R. H. Sweetser, Columbus, Ohio.—I visited this plant one very rainy day and found that the coal entering the tipple was much wetter than the coal going out; there was practically no drip out of the cars of coal as shipped.

Pulverized Coal as Fuel for Copper-Refining Furnaces

By E. S. Bardwell* and Roy H. Miller, Great Falls, Mont.

(Salt Lake City Meeting, September, 1925)

During the period extending from May, 1922, to September, 1923, the copper-refining furnaces of the Great Falls Reduction Department of the Anaconda Copper Mining Co. at Great Falls, Mont., were operated exclusively with pulverized coal as fuel. The installation was made first as an experiment. It proved successful from the start and was abandoned solely because fuel oil from the Sunburst field became available at a price sufficiently low to be more desirable from an economic standpoint. The equipment required for pulverized coal has, however, been kept intact and is available for immediate use should the oil supply fail.

Inasmuch as pulverized coal had been used on two large reverberatory smelting furnaces and was, at the time we were considering its application to the copper-refining furnaces, being used on the Wedge furnaces roasting zinc concentrate for the electrolytic zinc plant, we were equipped with a coal-pulverizing plant of ample capacity to supply pulverized coal for the furnace refinery. The only problem was to devise suitable means for transporting pulverized coal from the pulverizer plant, located in the smelter building, to the refinery about 1000 ft. distant and about 150 ft. above the level of the pulverizer plant. When studying this installation, it should be borne in mind that the layout represents an adaptation suited to the existing conditions at this particular plant. The installation was entirely satisfactory and uniformly dependable but might be modified to advantage from an operating point of view if an entirely new installation were being planned.

Fuel Supply

The use of the local coals, high in ash and sulfur, was abandoned some time before coal-dust firing was attempted. Local coals from the Great Falls field carrying an average of 4 per cent. sulfur and 22 per cent. ash were used with fair success in the small furnaces in use prior to 1916. In the 250-ton furnaces, however, the time lost in grating and removing clinker from the fireboxes, as well as the difficulties incident to the high-sulfur content of the fuel, became serious and recourse was had to the

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higher grade bituminous coals of Wyoming and Utah; more particularly the Diamondville coal from Wyoming and the Sunnyside coal from Utah. Both of these coals are low in ash and sulfur and both work well when burned in the pulverized form.

That these two fuels were nearly on a par, as regards suitability for the purpose to which they were being put, is shown by the following analyses, which represent the average of a large number of cars received during the first six months of 1923:

Kind of Fuel	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	B.T.U. Dry	B.T.U. Wet
Utah	5.0	33.6	53.0	8.4	0.8	12,757	12,190
Wyoming	5.4	36.0	50.9	7.7	1.1	12,689	12,010

Coal-pulverizing Plant

The coal-pulverizing plant is located in the main smelter building some distance from the furnace refinery. The coal, received in standard-gage railroad cars is dumped into one of two bins delivering directly to a 24 by 30 in. Jeffrey toothed-roll crusher. The product of the crusher goes, by gravity, to a 30-ton receiving bin. Coal is drawn from this bin on to an 18-in. belt conveyor, which delivers directly to an A-12 Ruggles Cole dryer. This dryer is fired with coal that has passed the crusher. When drying Utah or Wyoming coal, to avoid danger of fire, the temperature is carefully regulated so as not to exceed 50° C. at the exhaust-fan inlet; the average temperature at this point, when the dryer is in operation, is 40° C. A recording thermometer is provided to insure the maintenance of this condition at all times. The coal, as received, averages 5.5 per cent. moisture; after drying and pulverizing, 2.75 per cent. Under these conditions the capacity of the dryer is 25 tons per hour.

The coal discharged from the dryer goes, by gravity, to a 20 by 34 in. Jeffrey disintegrator, crushing down to 1/4 in., and then, by a screw conveyor, to a manganese-steel bucket elevator. This elevator delivers, through a series of 12-in. screw conveyors, to six 100-ton storage bins located directly over six Raymond five-roller pulverizers. Each pulverizer is separately driven by a 75-hp. motor and has a capacity of 3 tons per hour, when crushing Utah or Wyoming coal. The Raymond pulverizers are operated on a vacuum system; an exhaust fan placed above each mill pulls the coal from the mill as soon as it reaches the required degree of fineness. These fans discharge into separate cyclone collectors, which partly separate the coal and air. The pulverized coal from the collectors is delivered to screw conveyors; the air returns to the Raymond mills. The excess dust-laden air from the first collectors is passed through auxiliary cyclone collectors vented to the open air.

The dryer is provided with a cyclone-dust collector vented into the dryer stack. All conveyors, bins, and elevators are properly vented to the atmosphere, making it possible to keep the plant clean and free from accumulations of inflammable dust. The total loss by dusting has been found, by actual measurement, to amount to 0.6 per cent., or 12 lb. per wet ton of coal handled.

Screen sizing tests made on the coal delivered to the furnace refinery show 96 per cent. through 100 mesh and 80 per cent. through 200 mesh.

Transportation of Pulverized Coal

For the purpose of transporting the coal from the pulverizer plant to the refinery, two special tank cars were designed, as shown in Fig. 1. These cars were designed to hold 42 tons each of pulverized coal. They

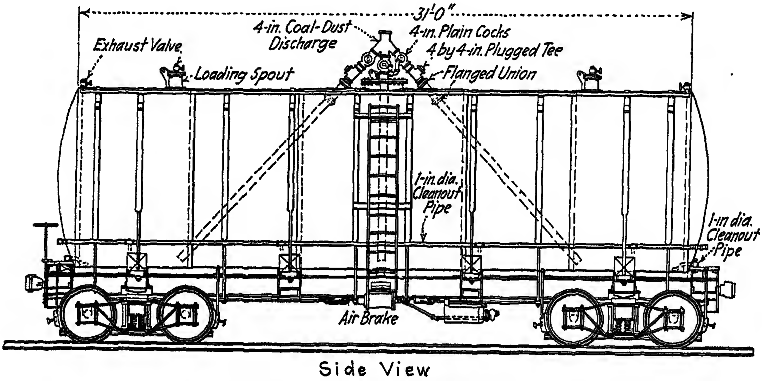

Fig. 1.—Coal-dust tank car; capacity 42 tons.

were standard-gage cars and were spotted for loading and unloading by the local electric tram, thus rendering the operation independent of the railroad switching service.

Each car is provided with three loading spouts, or openings, for filling. In practice the center opening only was used, as the coal dust charged through the center opening distributed itself very well, filling the car almost as well as if the other two openings had been used. When the car was spotted for loading, the exhaust valves in the ends were opened and the coal loaded through a canvas pipe receiving coal from the mills through a screw conveyor.

As soon as the car had been spotted for unloading, the delivery line was connected at the flange marked "4-in. coal discharge," and high-pressure air connections were made at the center manhole cover. All connections having been made tight, air was turned into the receiver and the pressure in the car allowed to build up to 40 lb., as indicated by a gage on the air line. As soon as this pressure was attained, the air valve

was closed and the center discharge valve opened allowing coal to be forced to the bins over the furnaces. The bins were 40 ft. above the level of the car and 165 ft. from the point of unloading. The pipe through which the pulverized coal was forced was 4 in. inside diameter. No trouble was experienced through this pipe becoming choked. When the pressure fell to 20 lb., the discharge valve was closed and the air valve opened until the air pressure again reached 40 lb., when the air valve was closed and the discharge valve was again opened. This procedure was repeated until three such blows had been made. The same procedure was then followed in unloading each end of the car. When making the last blows, air was turned into the 1-in. cleanout pipe, so as to stir up the coal in the bottom of the car in order that as much of it as possible might be carried over into the receiving bins. The receiving bins were covered at the top and vented through woolen bags attached to thimbles set in the covering of the bin and suspended from a beam overhead. The average load delivered to the furnace refinery in these cars was 38.4 tons, of which 12.3 tons remained in the cars after unloading. In other words, the net capacity of these cars handled as above described was 26 tons of pulverized coal.

Furnaces

The copper refinery at Great Falls is equipped with two furnaces having hearths 45 ft. long. One of these furnaces has a hearth 13 ft. 9 in. wide; the other is 18 in. wider. These furnaces have silica-brick bottoms, 32 in. thick, arranged for air cooling. Each furnace is equipped with a Sterling waste-heat boiler generating about 400 b.hp. The furnace stacks are 5 ft. in diameter and 142 ft. in height, measured from the working-floor level. No trouble was experienced from lack of draft.

When the change was made from grate firing to pulverized coal, the coal hoppers directly over the furnace fireboxes were used as storage for pulverized coal. In order to avoid fires in these bins, the bottoms were changed so as to eliminate dead-storage space; this change also tended to prevent hanging up the contents of the bins and consequent sudden rushes of dust to the burners. The burner pipes at first were introduced through openings in the firebox wall, it being thought that something in the way of a combustion chamber was necessary; it was soon discovered that no combustion chamber was required. The fireboxes were accordingly dismantled and the burner pipes introduced through openings directly over the bridge wall.

Three burners of the Warford type were used for each furnace; the arrangement of the burners is shown in Fig. 2. The screw conveyors supplying coal to the mixing chambers of each set of three burners were operated through bevel gears from a single-drive shaft, the speed of which was controlled by a Reeves speed-changing transmission. Primary air

for all six burners was supplied by one of two special 48-in. high-pressure Sirocco blowers, each driven by a direct-connected 75-hp. motor running at 1800 r.p.m. The primary air was delivered to the bustle pipe at 13 oz. pressure. Secondary air, in regulated amount, was drawn into the mixing chambers of the burners as well as around the burner pipes. When less than three burners were required, coal and air could be shut off from one or more burners. When heating the furnace, the burner pipes were removed from burners not required and the burner openings luted up.

Except for the removal of the fireboxes, no changes were made in the furnace design when the change was made from grate firing to coal-dust firing. The uptake flue leading to the waste-heat boilers has a minimum opening 2 ft. 8 in. by 3 ft. 5 in. This has given good results

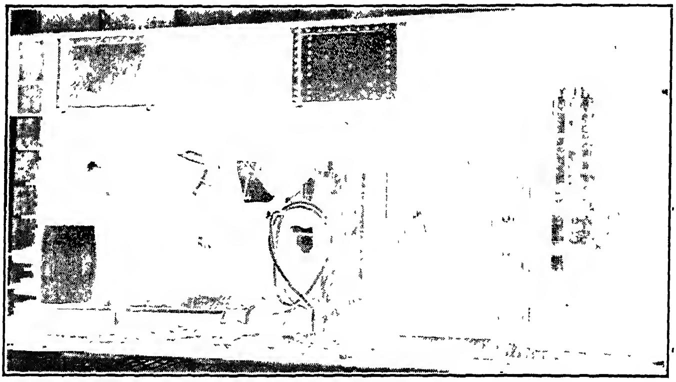

FIG. 2.—ARRANGEMENT OF WARFORD BURNERS.

not only with grate firing and pulverized coal, but has proved ample when the furnaces are burning oil, as at present. The draft, when using pulverized coal, was approximately 0.35 in. of water measured in the uptake flue leading to the waste-heat boilers.

EFFECT OF PULVERIZED COAL ON FURNACE OPERATIONS

An operator considering a change from grate firing to pulverized coal is desirous of knowing what economies may be brought about by the change. The use of pulverized coal at Great Falls enabled us to make our cycle with reasonable certainty every 24 hr. We were even able to make such minor repairs to flues and jambs as did not necessitate a shutdown and still take out seven charges weekly. A loss of 1 or 2 hr. on one day caused by such a repair could be caught up and the furnace brought around on schedule again within a day or two, at the most, and without any material sacrifice of production.

The saving in time, as compared with grate-firing, is brought about: First, by the saving of time required in grating and removing clinkers

from the firebox; second, by reason of the fact that when we were ready to start charging, the burners could be adjusted so as to maintain the heat in the smelting chamber to a much greater extent than is possible when grate-firing; third, during the melting period, conditions of combustion were far more uniform than was previously the case. Not only did this lead to a shorter melting period, but, due to the ability to maintain an oxidizing atmosphere in the furnace at all times while melting down, the rabbling period was materially shortened. A typical cycle of operations was: Charging, 2½ hr.; melting, 7½ hr.; rabbling, 6 hr.; poling, 3 hr.; and tapping, 4⅔ hr. During the melting period, the flue gases contained approximately 16.0 per cent. CO_2 and the gases entering the

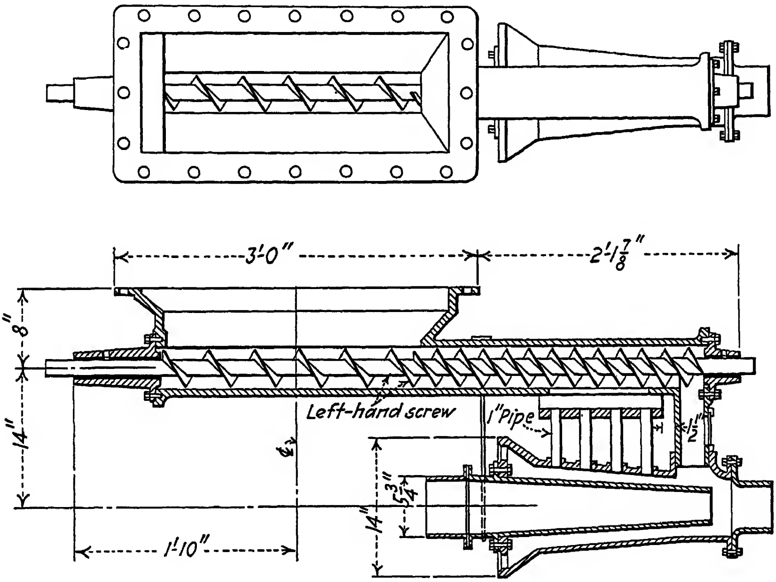

FIG. 3.—CONSTRUCTION OF WARFORD BURNERS.

stack after passing the waste-heat boilers were found to have a temperature of about 650° F. Inasmuch as the theoretical CO_2 content of the gases for a condition of perfect combustion is 17.0 per cent. for the wet gas, or 18.4 per cent. excluding moisture, we consider that the operation was highly efficient.

The use of pulverized coal also resulted in a considerable saving in fuel cost. The best fuel ratio we were able to maintain for an entire month, using coal with grate firing, was 4 tons of copper per ton of coal. With the same quality of coal used in the pulverized form, we obtained a ratio of 6.8 tons of copper per ton of coal over a 6 months period. This represents a saving of 41 per cent. in the coal bill, assuming that the cost of pulverizing is offset by a reduction in labor at the furnace and elimina-

tion of the cost of handling and disposing of ashes. A further saving was, however, realized, for when coal was burned on grates, lump coal only could be used; whereas when the coal was used in the pulverized form, slack coal, costing 20 per cent. less, could be used. The net result was a reduction of from 40 to 50 per cent. in the fuel cost per ton of copper.

How does the amount of slag produced per ton of copper, when firing with pulverized coal, compare with the amounts produced when grate-firing or using oil fuel? When making this comparison, inasmuch as the relative amounts of scrap being produced during the several periods under consideration differ for reasons entirely foreign to the subject we are considering, it has seemed best to eliminate the scrap entirely and base the figures on tons of slag produced per ton of copper melted and refined. On this basis per 100 tons of copper, using coal on grates, we made 4.2 tons of slag containing 2.04 tons of copper; using pulverized coal, we made 5.37 tons of slag containing 2.11 tons of copper; and using oil, we made 3.91 tons of slag containing 1.77 tons of copper. In other words, when using pulverized coal, 28 per cent. more slag was produced than when grate-firing, which slag contained, however, only 3.4 per cent. more copper than was the case when grate-firing. As compared to oil-firing, pulverized coal produced 37 per cent. more slag containing 19.2 per cent. more copper. The slag production for oil has been reduced somewhat since these figures were compiled, by improved methods of operating. Conditions under which the plant is operated and the grade of refractories obtainable account largely for the heavy slag production. The figures, however, show the comparative amount, and copper content, of slag produced under the several methods of firing. The amount of slag produced when pulverized coal is used depends largely on the fineness of the coal. From a purely operating standpoint, the coal can hardly be too fine. From an economic standpoint, however, the greatly increased cost of extreme fine grinding offsets any saving that might be realized in cost of slag treatment. The amount of ash settling in flues and stack was a small percentage of the total ash; this ash was removed at intervals of two to three months.

The quality of the copper produced during the period when pulverized coal was being used was uniformly excellent; the sulfur in the refined product seldom exceeded 0.002 per cent. No fear need be had as to the quality of the product as long as coal containing less than 2 per cent. sulfur is used and flues are properly proportioned so as to avoid an actual pressure of gases in the furnace above that of the air outside.

Although pulverized coal was used continuously for about 16 mo., it is not possible to decide, from the data on hand, whether any actual decrease in cost of keeping furnaces in repair was realized. A study of such data as we have, however, seems to indicate that the cost of repair was no greater than when grate-firing was in vogue. If the last months

during which coal was being burned on grates is compared with the first 6 mo. in which pulverized coal was used a decrease of 10 per cent. is shown in the cost of furnace repairs in favor of pulverized coal.

Conclusions

Bituminous coal running less than about 8 per cent. ash and 2 per cent. sulfur can be used in the pulverized form in connection with the melting and refining of cathode copper. The prime requisites are fine grinding and proper furnace and flue design.

The use of pulverized coal replacing coal of similar grade at Great Falls resulted in a saving of at least 40 per cent. in actual fuel cost, with some additional saving in labor. Oil may be regarded as the ideal fuel for the furnace-refining operation, both from the point of view of ease in handling and storing, and from the efficiency realized in its combustion. Pulverized coal is a close competitor, and any great increase in oil price would lead to the resumption of its use in our furnaces.

Selection of Coals for the Manufacture of Coke

By H. J. Rose,* Pittsburgh, Pa.

(New York Meeting, February, 1926)

Coke Production

Sixty-five million net tons of coal were carbonized in the by-product and beehive coke ovens[1] of the United States during 1924. This tonnage represented 13.4 per cent. of the bituminous coal which was mined in that year. Table 1 shows production and disposition.[2]

Table 1.—*Production and Disposition of Oven Coke in* 1924

	By-product	Beehive	Total
Coal charged into ovens, tons	49,061,339	15,914,310	64,975,649
Coke produced (exclusive of screenings and breeze), tons	33,983,568	10,286,037	44,269,605
Average yield of coke, per cent	69.3	6.46	68.9
Disposition of coke, per cent.:			
Blast furnace	82.5	82.8	82.6
Foundry	4.6	13.3	6.6
Domestic	8.3	1.4	6.7
Water gas manufacture and all other uses	4.6	2.5	4.1

Cokes having somewhat different combinations of chemical and physical properties are desired for the various uses listed above, and these properties depend primarily upon the types of coal used. American coals have a wide range of purity and coking quality, and the chief subject of this paper will be the selection of coals for the manufacture of blast furnace, foundry, domestic, and water-gas coke, in by-product coke ovens. It will obviously be necessary to include a discussion of those physical and chemical characteristics that are generally considered essential or desirable for each type of coke.

Suitable Coals

The examination and development of new sources of coking coal are important because of the irregular and limited distribution of high-grade

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[1] About five million additional tons of coal were carbonized in gas retorts.

[2] Adapted from U. S. Bureau of Mines report No. 425, Sept. 5, 1925.

coals of established coking quality, and the rapid rate at which our reserves of such coals are being depleted for various purposes. Great advances in the design and construction of by-product coke ovens have recently made possible the use of coals which were hitherto considered unsuitable for coking. Therefore, a description of several methods for determining the quality of coke that can be produced from a given coal, is appropriate, also a brief discussion of the relations between the chemical composition and coking properties of coal. The characteristic behavior of certain coals when heated, which makes them amenable to coking, will first be described.

Behavior of Coking Coals When Heated

Coke is the coherent cellular residue from the destructive distillation of certain bituminous coals. When a small sample of such a coal is heated at a uniformly increasing temperature in the absence of air, it will exhibit a definite fusing or softening temperature, usually occurring

Fig. 1.—Cell Structure of By-product Coke Made from 34 Per Cent. Volatile Matter, High-oxygen Coal, No. 6 Seam, Franklin County, Illinois. × 25.

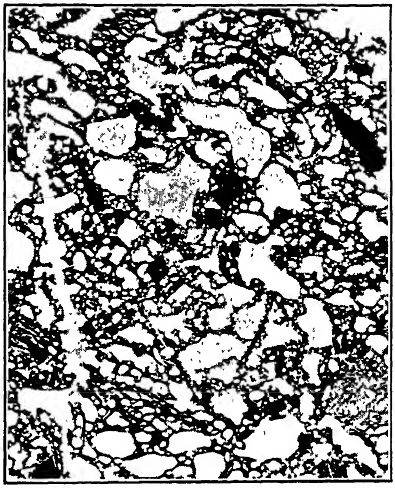

Fig. 2.—Cell Structure of Beehive Coke from 34 Per Cent. Volatile Matter, Washed Coal, Las Animas County, Colo. Cells Are Filled with White Composition and Only the Cell Walls Are Visible. × 4.

between 300° and 400° C., and will become pasty or semifluid. Gases of decomposition begin to appear in appreciable quantities at the fusing temperature, or slightly higher, and are soon evolved copiously, causing the mass to become cellular and more or less swollen. The viscosity of the fused and pasty coal rapidly increases as decomposition proceeds, and the hot mass becomes substantially rigid before a temperature

of 450° C. is reached. However, it is continually subject to further changes in structure, due to devolatilization and shrinkage, and these changes are still in evidence after the coke has reached a red heat. Gas evolution continues at a rapid rate long after the initial rigidity has been attained.

Such in brief, is the behavior of all good coking coals when heated. Poorly coking coals react similarly in a varying but less degree, and the temperatures of initial gas evolution vary considerably. In fact, the typical cellular structure of coke[3] is caused by the evolution of gas bubbles in a mass of fused coal that is stiffening through thermal decomposition (Figs. 1 and 2.) Non-coking coals do not fuse and become pasty under the influence of heat.

Coke Formation in a By-Product Oven

We have been considering the case of a small sample of coal heated at a uniformly increasing temperature, but when coking coal is charged into a red-hot chamber, the resulting conditions are considerably more complex. In the by-product coke oven, the coal which comes into contact with the hot walls is rapidly carbonized, and a narrow pasty layer called the "plastic zone," or "pitchy seam," is formed parallel to each heating wall. As carbonization proceeds, these layers move towards the center of the oven at an average rate of about 0.5 to 0.65 inch per hour. The plastic zones consist of fused coal with tarry or pitchy material from coal that is being distilled. The thickness and nature of these zones depend on the kind of coal used, the degree of pulverization, rate of coking, etc. With good coking coals, especially when they have been well crushed or pulverized, there can be little doubt that the plastic zones are substantially impermeable to gases during most of the coking period.

There is a sharp temperature gradient in the vicinity of these zones. Most of the gases and vapors are liberated on the hot side of the plastic zone and in the adjacent low-temperature coke, and are forced by their own pressure towards the oven walls, where they stream upwards, either along the walls or through shrinkage cracks in the coke. The rich gases are cracked to some extent by contact with hot surfaces, and they accordingly deposit on the coke a silvery carbon or hydrocarbon film often referred to as a graphitic carbon deposit.

To show the progressive nature of coke formation, three boxes of coal were charged at intervals into an otherwise empty coke oven, and were discharged together before the plastic seam had quite reached the center of the box charged first (shown at the left in Fig. 3). The boxes were immediately quenched, the tops cut away, and the upper layers of coke removed to show the progress of coking.

[3] H. J. Rose: The Study of Coke Macrostructure. *Ind. & Eng. Chem.*, (1925), **17**, 895.

FIG. 3.—BOX TESTS MADE IN EMPTY COKE OVEN TO SHOW THE PROGRESSIVE NATURE OF COKE FORMATION.

When the coking process is well under way, a variety of solid products exist in the oven. Adjacent to the walls there is a layer of high-temperature coke which has already begun to shrink and fracture somewhat from the intense heat, and which is acquiring a silvery surface deposit. This high-temperature coke is continuous with, and grades imperceptibly into lower and still lower temperature coke, until the plastic zone is reached. The plastic zone is usually ¾ to ¼ in. thick, and on the hot side is cellular and merges rather indistinctly into low-temperature coke. The cool side of the plastic zone, where the coal is just starting to fuse, is very sharply defined. Individual fragments of coal on the border line are seen to be fused at one end, and little changed at the other end. The coal in the interior of the oven and shielded by the plastic zones remains unchanged and at a temperature little above that of boiling water for many hours, until it is finally reached by the slowly moving pasty layers. In actual practice, coking is usually continued not only until the plastic zones have met, but until the coke at the center of the oven has attained a red heat. A division line is present where the two plastic zones have come together, and the coke mass breaks apart along this plane, so that individual coke pieces are but half as long as the oven width.

Definition of Coking Coal

The commercial value of coke depends upon its purity, and upon such factors as size, strength and structure. These physical characteristics are determined not only by the kind of coal used, but also by the degree of pulverization and moisture content of the coal, the amount, nature and size of impurities such as slate, the temperature of the oven walls, dimensions of the oven and other variables, all of which affect the coke at some stage of its formation. Therefore, it is not surprising to find that the only thoroughly reliable way to determine the quality of coke that can be made from a coal under particular conditions, is to subject the coal to a full-scale oven test under the desired conditions.

For practical purposes, a coal may be classed as a coking coal if it will yield a merchantable coke when carbonized by a commercial method in an existing type of coke oven. In order to be merchantable, a coke must, in general, be of adequate purity, size, strength and structure, for the uses to which it is to be put. Furthermore, the coke from any coal or coal mixture carbonized in by-product ovens must be strong enough, and have sufficient shrinkage from the walls, to permit its discharge from the ovens without difficulty. Coal mixtures are commonly used at by-product coke-oven plants, and this practice has resulted not only in the production of superior coke, but in the extensive utilization of coals which would not be very suitable for coke manufacture, if used unmixed.

Determination of the Coking Properties of Coal

The problem of determining the coking quality of coal by test methods has received an amount of attention warranted by the great industrial importance of coke. It will not be practical to mention in detail the many methods that may be used as an aid to determining coking properties, but a few of the preferred methods will be briefly discussed, and their scope of usefulness indicated.

Oven Coking-tests

It has already been stated that full-scale oven tests form the only accurate and thoroughly reliable means of evaluating the coking properties of coal. In order to produce the best possible quality of coke in such tests, the condition of the coal as charged and plant operating conditions must often be varied. For these reasons, and because of the expense involved in such tests, it is important that they be carried out under competent direction, to obtain the best possible coke with the minimum number of tests. For a single charge in a full-sized by-product coke oven of the most modern type, 11 to 15 tons of coal are required, and as more than a single test is usually desirable, a carload of coal is the minimum quantity recommended for such a test.

Box Coking-tests

If coking quality of coals is doubtful or if it is suspected that the coke formed will not shrink sufficiently to permit ready pushing from the oven, it is advisable to make a preliminary test by embedding a specially designed metal box of the coal in a regular oven coal-charge. If the results are definitely unfavorable, the expense of a full-scale oven test is thus eliminated and possible interference with plant operation avoided. Box tests are also recommended when but little coal is available. (Fig. 3.)

When properly made box tests are of considerable value, and give to the experienced observer a definite idea of the character and coking possibilities of the coal tested. They cannot take the place of full oven tests when an accurate determination of the physical properties of the coke is required. Very erroneous ideas may be obtained from incorrectly made box tests.

Laboratory Coking Tests

Sometimes an approximate determination of the coking properties of a very small sample of coal must be made, for example, on part of a coal core from diamond drilling. An apparatus[4] devised by F. W. Sperr, Jr., permits a fairly satisfactory determination to be made on a sample

[4] U. S. Patent No. 1444567.

of about 2 oz. placed in a tube of such diameter that the coal occupies a length equal to half the width of a coke oven. Coking is begun at one end of the coal by means of a previously heated furnace, and the coal and furnace are moved with respect to each other at such a rate that the rate of progression of heat into a coke-oven charge is duplicated. An ingenious arrangement permits shrinkage or expansion under any desired load to be measured continuously throughout the test Much interesting and valuable information has been obtained with this apparatus.

The simplest of all coking tests is no doubt the standard volatile-matter determination in which a gram of powdered coal is suddenly heated to redness in a small platinum crucible. The resulting coke button will indicate whether the coal is non-coking, feebly coking or strongly coking, but almost every condition in this test is at variance with large-scale carbonizing practice.

Over a period of many years all of the above types of coking test have proved to be useful, full-scale oven tests or box tests being invariably used whenever practicable. Many other tests have been proposed and used by coal investigators, some of which have undoubted merit for their respective purposes. One such test, which with numerous variations is often used abroad, consists in mixing a definite weight of coal with successive and increasing proportions of carefully sized sand or carbon particles. In a typical test of this sort, the ratio of sand to coal in that mixture which, when coked, will just support a stated load without crushing and without forming more than a specified amount of fines, is taken as the coking index or sand-test number.

Correlation of the Chemical Composition and Coking Properties of Coal

The question "Can the coking properties of coal be predicted by chemical analysis?" is frequently asked. In spite of the many studies which have been made of coking coals, an affirmative answer cannot be given without reservations.

Use of Ultimate Analyses

The writer has made a graphic study[5] of the ultimate analyses of many hundreds of American and foreign coking coals, and has been particularly fortunate in having available the by-product yields for a large number of these coals, together with scores of complete records

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[5] H. J. Rose: Graphic Study of Coking Coals. (Unpublished.) Presented before the Gas and Fuel Section of the American Chemical Society, Washington, D. C. April 25, 1924.

of by-product oven coking tests. All coking coals appear to fall within fairly definite chemical limits, so far as their ultimate analyses are concerned, and it is the writer's experience that from the location of a coal within these limits, the coking properties and by-product yields may be predicted with reasonable accuracy, *if all essential data on the chemical and physical characteristics of the coal as it will be charged, and the operating conditions which will exist, are known.* Coke quality and by-product yields are greatly affected by many chemical and physical variables, and by operating conditions, and while the influence of each of these independent variables is more or less well understood, the difficulty of evaluating the effect of complex combinations can easily be appreciated. Experience indicates that such a task should be left to specialists who have had ample practical experience in testing a wide variety of coals.

The most rational scheme of studying coals according to their ultimate analyses is undoubtedly the use of trilinear coordinates, as proposed by Grout[6] nearly 20 years ago, and as amplified by Ralston[7] about 10 years ago. For this graphic method the ultimate analyses must be reduced to three variables which always total a constant amount. One method is to calculate the analyses free of moisture, ash, sulfur, and nitrogen, so that the sum of the remaining carbon, hydrogen and oxygen will always total 100 per cent., but obviously two or more items can be combined in any of the three variables. If a right-angled triangle is used as the basis of the coordinate system, then ordinary rectangular coordinate paper can be used by expressing two of the variables with the existing lines and by drawing in diagonal lines to express the third. Of course the third variable need not be indicated at all, as its numerical value can always be obtained by subtracting the sum of the other two variables from 100 per cent.

Seyler's Classification

Such a system would be the graphical equivalent of the coal classification proposed by Seyler[8] in 1900. Seyler, using ultimate analyses calcuated free from ash and sulfur, divides coals into named groups by means of limit percentages of carbon and hydrogen so selected that the coals in each group possess certain resemblances. This classification is used by British fuel investigators, and apparently agrees rather well with several graphic classifications of American coals.

[6] F. F. Grout: The Composition of Coals. *Econ. Geol.* (1907), **2**, 225.

[7] O. C. Ralston: Graphic Studies of Ultimate Analyses of Coals. *Tech. Paper* 93, U. S. Bureau of Mines (1915).

[8] C. A. Seyler: Proc. South Wales Inst. Eng. (1900), **21**, 483 et seq.; also The Chemical Classification of Coal, Fuel in Science and Practice. (1924), **3**, 15, 41, 79.

Use of Ratios

When the ultimate analyses of solid fuels are plotted on triangular coordinates, the analyses of wood, peat and all coals ranging from lignite to anthracite (cannel and alga coals being excepted) fall within a remarkably narrow band in one corner of the triangle. Fig. 4 (adapted from Ralston's paper) shows part of this band on which the location of the various ranks of coal are indicated. The stippling on this figure does not represent actual coal analyses, but simply indicates the limits within which most coal analyses will fall. The bituminous and semi-bituminous classes consist chiefly of coking coals.

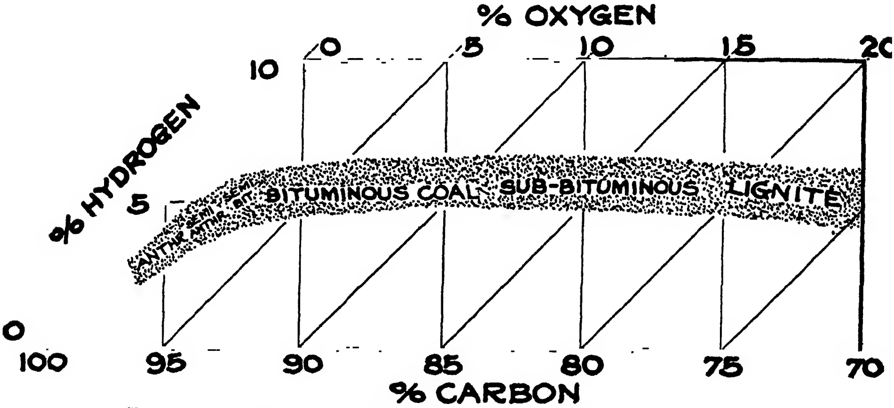

FIG. 4.—TRI-AXIAL DIAGRAM OF ULTIMATE ANALYSES OF COALS.

Many investigators have attempted to classify coals by the use of certain ratios such as Hydrogen:Oxygen, Hydrogen $- \frac{\text{Oxygen}}{8}$, Carbon:Hydrogen, and Carbon:Oxygen. To understand what these ratios mean when applied to coal is rather difficult. Figs. 5, 6 and 7 will help to illustrate their probable usefulness, also the obvious limitations of such ratios. Detailed discussion of these figures is beyond the scope of this paper, but it should be pointed out that owing to the sharp curvature in the band of coal analyses occurring at about 92 per cent. carbon, 5 per cent. hydrogen, and 3 per cent. oxygen, no linear equation can be adequate over the whole range of coals.

Hydrogen:Oxygen Ratio

Thus in Fig. 5, the lower H:O ratios cut across the band of lignite, sub-bituminous and bituminous coals (though much more obliquely than the oxygen lines), but for the coals of higher rank, the ratio lines run *parallel* with the coal band. Thus, a single H:O ratio line may pass through as many as four ranks of coal (as well as cannel coals, which are

not shown but would fall above the bituminous coals). The H:O ratio is particularly well known because of White's frequently quoted conclusions[9] on the limits of this ratio for coals that can be coked in beehive ovens. White pointed out that the relation between H:O ratio and coking quality was subject to exceptions for the coals of lower volatile-matter content.

Fig. 5 also permits the graphic estimation of the "available hydrogen" in coal, that is, the percentage of hydrogen in excess of that which would be required to combine with all of the oxygen present to form water. This index has long been known but is generally considered to be of doubtful reliability.

Carbon:Hydrogen Ratio

Fig. 6 shows that the C:H ratio lines are not very different in slope from the hydrogen lines, and apparently for most purposes the hydrogen percentage in pure coal would be just as satisfactory an index, and one simpler to use and understand. The C:H ratio has been chiefly used in districts where anthracite, semi-anthracite and semi-bituminous coals are produced, for which coals this ratio serves as a means of differentiation just as the hydrogen percentage would. For coals of lower ranks the ratio would appear to have a very limited use, as it does not distinguish between coals of widely different rank.

Carbon:Oxygen Ratio

Fig. 7 shows that the slope of C:O ratio lines is so nearly the same as that of the oxygen lines that the use of the oxygen percentage in pure coal would probably be just as satisfactory for most purposes.

The application of other ratios, such as $\frac{\text{Volatile carbon}}{\text{Carbon}}$, $\frac{\text{Fixed carbon}}{\text{Volatile matter}}$, are subject to well known limitations which need not be discussed here. A study of the shortcomings of the various ratios and indexes which have been proposed from time to time, indicates that it is not possible to classify coals adequately by means of single index numbers, but that a two-dimensional system or diagram is necessary. If ultimate analyses are to be used as the basis of classification, systems such as those of Ralston and Seyler are indicated as the most logical.

Tri-axial Diagram of Coking Coals of the United States

Fig. 8 represents the ultimate analyses of about 150 coals from 10 different States. They have been classed as "typical" coking coals in the sense that they represent a wide geographical distribution and range of chemical composition. Most of these coals have been used for

[9] D. White: The Effect of Oxygen in Coal, *Bull.* 29, U. S. Bureau of Mines (1911), or U. S. Geol. Survey *Bull.* 382.

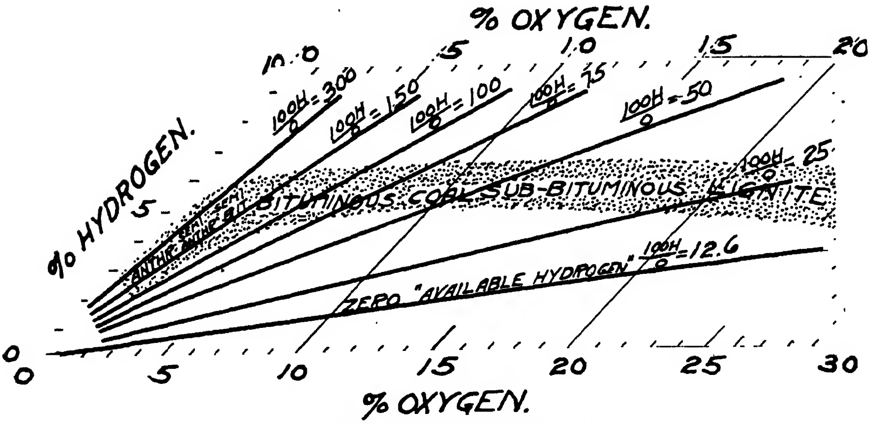

Fig. 5.—Hydrogen: Oxygen Ratios, Tri-axial Diagram.

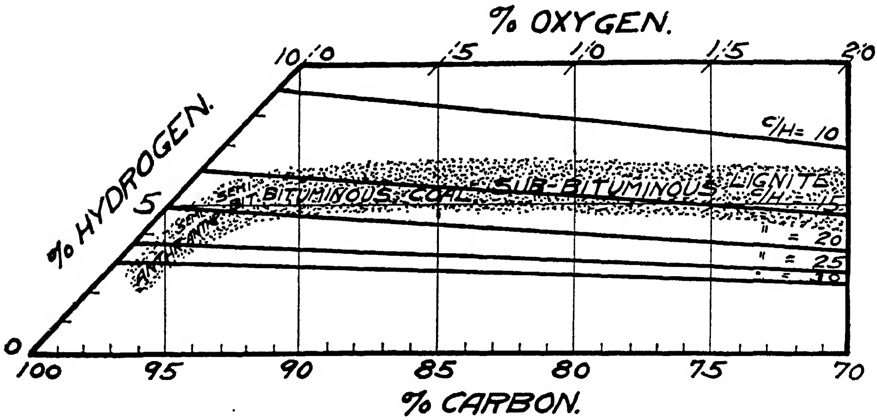

Fig. 6.—Carbon: Hydrogen Ratios, Tri-axial Diagram.

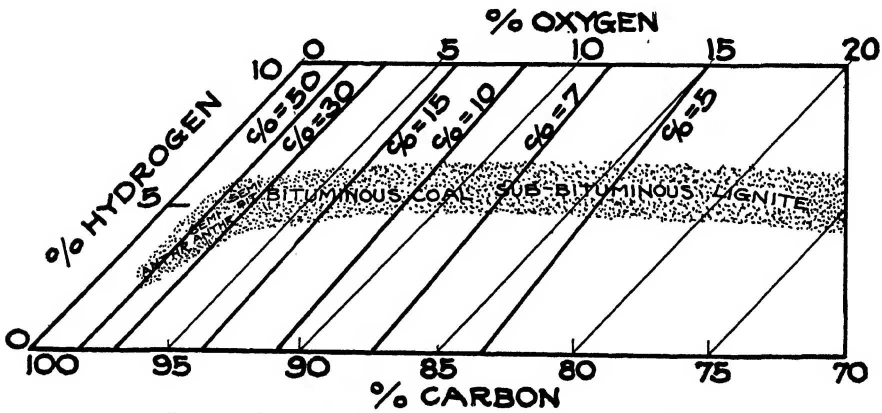

Fig. 7.—Carbon: Oxygen Ratios, Tri-axial Diagram.

by-product coking, either alone or in mixture with other coals. By combining Parr's "unit coal" correction[10] for ash and sulfur with the nitrogen content, the original ultimate analyses have been recalculated to the basis of Carbon + Hydrogen + Oxygen = 100 per cent. It should be noted that the vertical, or hydrogen, percentage scale of this chart has

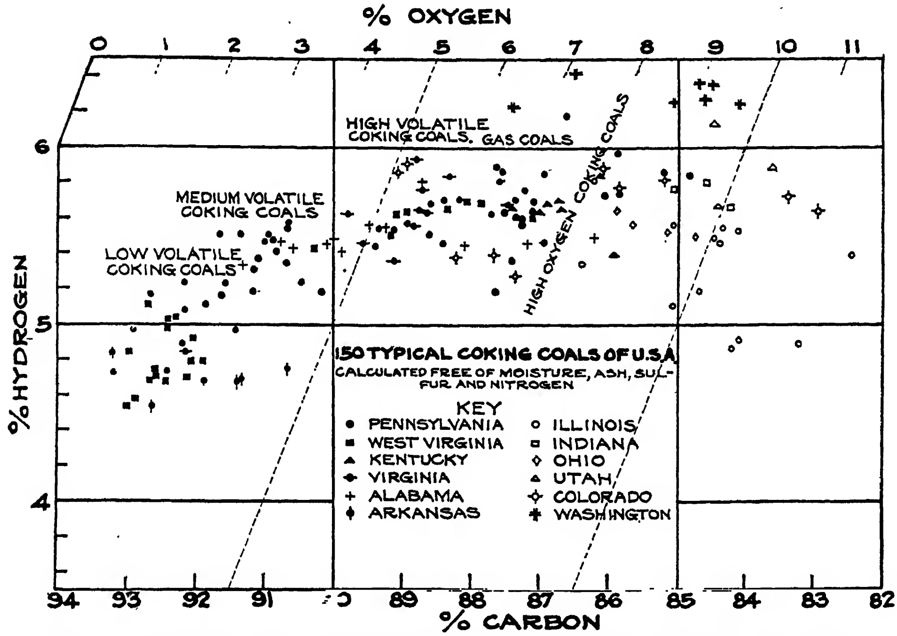

FIG. 8.—DIAGRAM OF THE ULTIMATE ANALYSES OF 150 COKING COALS OF THE UNITED STATES.

been expanded two and one-half times with respect to the horizontal or carbon percentage scale, in order to permit all points to be distinctly plotted. If the same scale had been used for both components, the band of analyses would have been correspondingly narrowed and more sharply defined.

CLASSIFICATION OF COAL BY VOLATILE MATTER CONTENT

The terms low, medium, and high-volatile coking coals, gas coals, and high-oxygen coking coals, which are used on Fig. 8, do not designate sharply defined classes. These and similar terms are used by by-product coke-oven operators in a relative sense, and the exact meanings intended by the individual operator may vary somewhat with the nature of the coals with which he is best acquainted, or which are available for his plant.

[10] S. W. Parr: The Classification of Coal. *Jnl. Ind. & Eng. Chem.* (1922), **14**, 919.

Low-volatile Coking Coal

In the northeastern section of the United States (where more than 80 per cent. of the by-product coke is produced), the term "low-volatile coal" is used chiefly to designate coals of less than 20 per cent. volatile matter, such as the Pocahontas and New River, W. Va., and Somerset County, Pa., type. For by-product oven use, such coals are mixed with high-volatile coking or gas coals in any proportion up to 60 per cent. or more (but usually from 10 to 30 per cent.), in order to increase the size and strength of the coke. This practice also results, of course, in an increase of the coke yield and decrease in the by-product yields.

Most by-product oven operators in selecting low-volatile coals prefer coals having a volatile-matter content about 16 to 18 per cent. (dry-coal basis), although a considerable quantity of coal that does not come within this range is used. True low-volatile coals are not charged in an unmixed condition into by-product coke ovens, because of their expanding properties when coked.

Medium-volatile Coking Coal

There appear to be no well defined, accepted limits for medium volatile coking coals, and usage of this term is known to vary. The writer's personal preference is to include in this class those coals ranging from about 22 to 28 per cent. volatile matter, dry basis. Such coals have certain coking characteristics which differentiate them from coals having distinctly higher or lower volatile-matter content. The medium-volatile coals, when coked without admixture, nearly always produce a large, handsome, blocky coke that is difficult to surpass in general physical characteristics.

Experience has shown that many of the coals in this class, particularly those having less than 24 to 25 per cent. volatile matter, are "neutral" *i. e.* practically non-shrinking, or even slightly expanding under many operating conditions, in which cases the coke would be difficult or impossible to push from the coke oven. If coking such coal in an unmixed condition in a by-product oven is desired, very careful and thorough tests are required to determine the conditions under which it may be satisfactorily used, and both the quality of the coal and plant operating conditions must be carefully watched to prevent possible trouble. Fig. 9 shows a wharf-full of coke, made from 21.1 per cent. volatile-matter coal from Washington, which was easily pushed from a 14-in. oven in 11 hr. 47 min. net coking time. Other coals of similar volatile-matter content have been successfully coked in the same type of oven.

High-volatile Coking Coal

Enormous quantities of coal ranging from 28 to 35 per cent. volatile matter, dry basis, are used in by-product coke ovens. Some of the largest plants coke such coal in an unmixed condition; many others use mixtures in which such coals are the principal components. High-volatile coking coal is produced from many seams and in many States, but the Pittsburgh seam coal from Pennsylvania and West Virginia is probably the most famous, because of the great extent of the seam and the large quantity of coking coal which is produced.

Fig. 9.—Low-volatile Coal Coked from 14-in. Becker Oven. Run-of-oven Coke on Wharf, Made from 21.1 Per Cent. Volatile Matter Coal Mined in Pierce County, Washington. This Coke Was Easily Discharged from the Oven. Net Coking Time, 11 Hr. 47 Min.

Recent improvements in coke-oven design have greatly increased the coking possibilities of straight high-volatile coals. Fig. 10 shows loaded cars of blast-furnace coke produced from 100 per cent. Pittsburgh seam coal with 34 per cent. volatile matter.

In the broadest sense, the term high-volatile coking coal includes both gas coals and high-oxygen coking coals, which are separately described on the page following.

Gas Coal

Gas coals may be defined as high-volatile coking coals suitable for the manufacture of city gas. The term was formerly restricted to those coals that had proved satisfactory for use in coal-gas retorts, but this distinction is losing its importance, because of the very extensive use of coke-oven gas for city supply.

FIG. 10.—LOADED CARS OF BLAST-FURNACE COKE MADE FROM 34 PER CENT. VOLATILE MATTER PITTSBURGH SEAM COAL, FAYETTE COUNTY, PA. COKED IN 14-IN. BECKER TYPE OVENS; COKING TIME, 11 HR. 40 MIN.

More than half of the total coal gas distributed through city mains in the United States is now obtained from by-product coke ovens, and the proportion is rapidly increasing, as the following figures will indicate. In 1924, more than 65 billion cu. ft. of coke-oven gas were distributed through city mains. During the fiscal year 1924, new coal-gas capacity, due to Becker type ovens completed or contracted for and including coal gas released by building gas producers at older plants, amounted to 25 *billion cu. ft.* This enormous increase in annual capacity includes only those installations which will actually supply gas for city use.

Gas coals are characterized by a high yield of B. t. u.'s in gas per pound of coal, and they have a volatile content ranging from about 33 to 38 per cent., dry basis. When carbonized in by-product ovens, they may be used alone, or with an admixture of low-volatile coal to improve coke quality. The most widely known gas coals are produced in West Virginia, Kentucky, Pennsylvania and Virginia.

Gas coals should be low in sulfur, about one-quarter or one-third of which appears in the gas as hydrogen sulfide, which must be completely, removed before the gas is sold for city use. Organic sulfur compounds, such as carbon bisulfide, also occur in the gas, and are not removed by any purification process commercially used in this country. This limits the permissible sulfur content of gas coals, as city regulations commonly specify that the total sulfur content of the gas shall not exceed 30 grains per 100 cu. ft. A 30-grain total sulfur specification is sufficiently liberal in most localities, and seldom interferes with the use of a coal which is considered satisfactory from the standpoint of hydrogen sulfide in gas, and sulfur in coke.

Most large gas companies require that gas coals for their use shall contain less than 1.25 per cent. sulfur, and a large amount of gas coal used in this country has a sulfur content well under 1.00 per cent.

High-oxygen Coking Coal

Last to claim our attention are the high-oxygen coking coals, which contain about 8 to 11 per cent. oxygen in the pure coal substance, and about 32 to 42 per cent. volatile matter, dry basis. When such coals are carbonized, they tend to give a smaller, weaker and more fingery coke than any of the coals previously considered. Many of them give under some conditions, a more or less pebbly, friable product which cannot be termed good coke, and which has little commercial value. Gas is produced in good volume, but is only fair in quality, because of its large content of carbon dioxide and carbon monoxide and a reduced methane percentage. Coals of this type usually contain considerably more moisture than coals of higher rank and, furthermore, they yield a higher percentage of water of decomposition when coked.

High-oxygen coals require a distinctly greater amount of heat for carbonization than other coals, and a high coking temperature is required for best results. For these reasons both a higher coking temperature and slightly longer coking time are usually employed for carbonizing such coals.

The beneficial effect of high temperature and rapid heat penetration is well shown by Figs. 11 and 12, representing longitudinal sections of two very different grades of coke made from the same coal in different

parts of the same coking-test box. For this test weathered Illinois slack coal, classed as non-coking, was placed in a narrow sheet-steel test box and coked in an otherwise empty oven. Under these conditions, the test box was at once subjected to heat from all directions, and coking proceeded very rapidly at the angles and corners of the box, where heat passing through two or three faces was concentrated upon a limited amount of coal. Heat passing through the sides of the box carbonized most of the coal in the normal manner, and Fig. 11 shows the weak, pebbly and very inferior coke formed in this way. In the angles of the boxes,

FIG. 11.—INFERIOR PEBBLY COKE FORMED FROM WEATHERED ILLINOIS SLACK COAL, COKED AT NORMAL RATE (IN NARROW BOX).

FIG. 12.—HARD TRUE COKE FORMED FROM WEATHERED ILLINOIS SLACK COAL. COKED AT FAST RATE (IN CORNER OF SAME BOX AS COKE SHOWN IN FIG. 11).

however, and particularly in the corners, a true, hard, bright coke was formed, as is shown by Fig. 12. This example will serve to illustrate a pitfall into which inexperienced experimeters often fall, when they carbonize small quantities of specially treated coal in very hot laboratory furnaces, and "discover" that hard coke can be made from poorly coking coals by their process.

At Provo, Utah, a battery of by-product coke ovens is producing blast-furnace coke from 100 per cent. Carbon County, Utah, coal.[11] This coal contains more than 10 per cent. oxygen in the pure coal substance, and over 40 per cent. volatile matter, dry basis. It is undoubtedly the lowest

[11] C. T. Keigley: The New By-product Coke Plant of the Columbia Steel Corporation. *Min. and Met.* (1925), 6, 422.

rank coal being carbonized in America, yet a very satisfactory furnace coke is being obtained. The coke is used to produce basic and foundry iron in a blast furnace 83 ft. high with a hearth diameter of 15 ft. P. W.

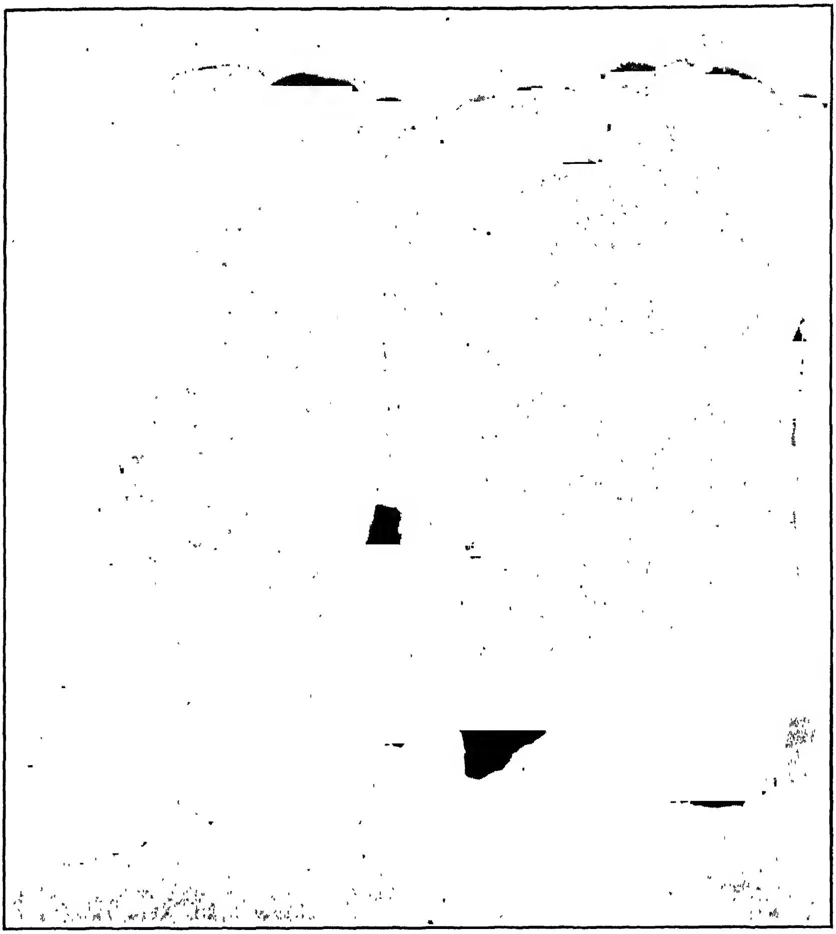

FIG. 13.—LONGITUDINAL SECTION OF REGULAR PLANT COKE PRODUCED FROM 40 PER CENT. VOLATILE MATTER, HIGH-OXYGEN COAL, CARBON COUNTY, UTAH, IN 14-IN. BECKER TYPE OVENS.

Jackson, blast-furnace superintendent at this plant, stated in a recent article:[12] "The March production of 11,582 tons of iron with a low fuel consumption per ton of iron, compares favorably with any practice

[12] P. W. Jackson: Blast-furnace Plant of the Columbia Steel Corporation. *Min. and Met.*, (1925), **6**, 420.

in the country and the limit of progress has not yet been reached." Fig. 13 illustrates longitudinal sections of the regular plant coke produced at Provo.

Many high-oxygen coals have been very successfully coked in the same type of oven. Fig. 14 shows run-of-oven coke from Illinois coal, lying on the coke wharf.

The coking properties of all coals are adversely affected by weathering (oxidation), and it is particularly essential that high-oxygen

FIG. 14.—RUN-OF-OVEN COKE FROM 35 PER CENT. VOLATILE MATTER HIGH-OXYGEN COAL, NO. 6 SEAM, JEFFERSON COUNTY, ILL., ON WHARF. COKED IN 14-IN. BECKER TYPE OVENS.

coals be carbonized in a freshly-mined condition in order to obtain best results.

The best known high-oxygen coking coals are those in Indiana, Illinois and Utah, but some are also found in Ohio, western Kentucky, Colorado, Washington, and other States.

Volatile Matter and Calorific Value for Coal Classification

It will have been noted in the preceding paragraphs that volatile matter alone is not a safe criterion for classifying coking coals, as the volatile-matter limits of high-oxygen coking coal, gas coal and high-

volatile coking coal overlap considerably. This may be explained by the fact that although hydrogen and oxygen are both responsible for the formation of volatile matter, they act very differently in affecting the quality of coke and by-products obtained. Thus a gas coal of 34 per cent. volatile matter may have excellent coking properties and give a high yield of rich gas, whereas a high-oxygen coal also having 34 per cent. volatile matter may give a very poor coke under many conditions, and yield a gas of rather low heating value.

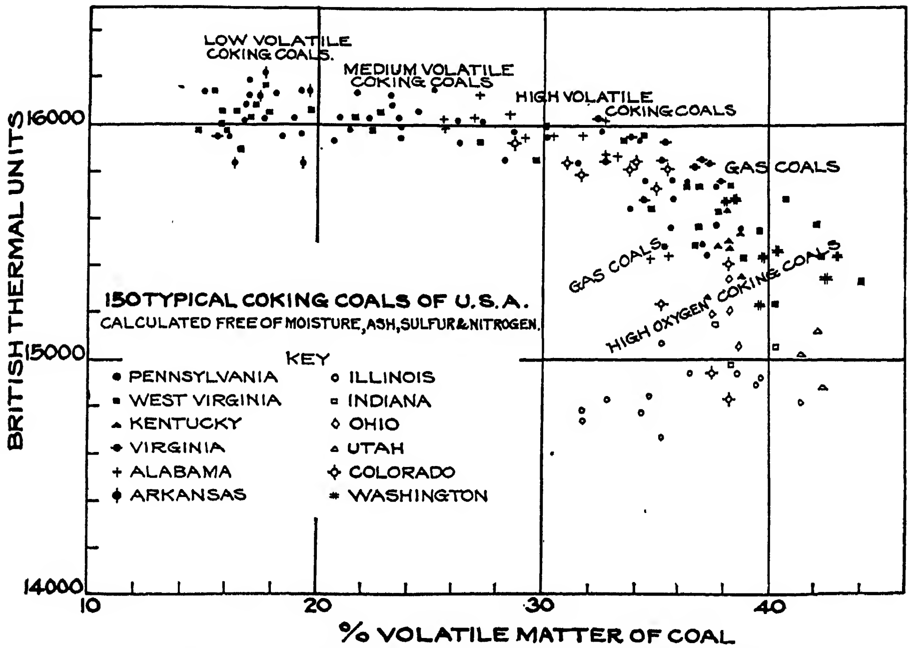

Fig. 15.—Curve Showing Volatile Matter and Calorific Value of the Same 150 Coking Coals of the United States Ultimate Analyses of Which Are Plotted in Fig. 8.

When coking coals are classified by means of their percentages of volatile matter, it is highly desirable to determine either their oxygen content or their calorific value. In fact, it is necessary to do so for the higher-volatile coals from unfamiliar mining districts.

It has long been known that the calorific value of a coal closely depends upon its ultimate analysis. Although we are primarily interested in the oxygen content of a coking coal, the calorific value is often more accurately determined than the oxygen figure, and moreover, it is frequently available when the latter figure has not been obtained. Parr[10] has proposed a classification of coals based upon the volatile matter and calorific value of pure or "unit coal." Fig. 15 represents the same 150 coking

coals that are shown in Fig. 8. It will be noted that the curves are very similar, and that coals fall into the same classification when plotted by the two methods. The vertical scale in Fig. 15 is more compressed than that in Fig. 8, thus causing the points to fall within narrower limits along that dimension. The graph clearly shows how high-volatile coking coals, gas coals and high-oxygen coking coals overlap in volatile-matter content.

In addition to the use of Parr's special "unit coal" correction for ash and sulfur, the analyses in Fig. 15 have been calculated free of nitrogen, and calorific values therefore average about 200 B. t. u's. higher than they otherwise would.

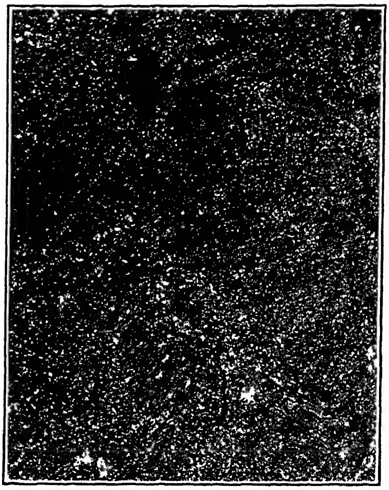

Fig. 16.—Longitudinal Section of Box-test Coke. Made in By-product Oven from 17 Per Cent. Volatile Matter Coal, Pocahontas No. 3 Seam, McDowell County, W. Va. × 1.

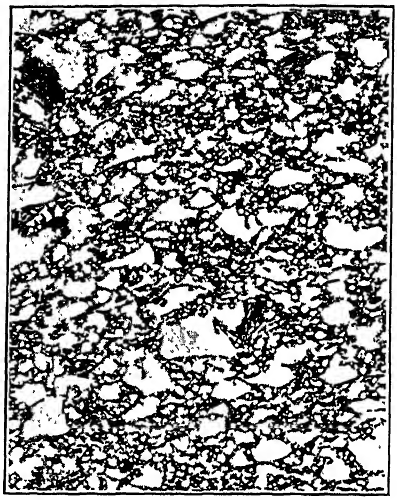

Fig. 17.—Longitudinal Section of Pocahontas 72-hour Beehive Coke, McDowell County, W. Va. Cells Filled with White Composition. × 4.

Coal Classes and Coke Structure

The following generalizations will prove useful, although they are subject to many exceptions due to the effect of independent variables.

Small-celled Coke

Low-volatile coals when coked alone, give large, tough blocky coke pieces which are vastly different in general appearance from the usually slender or fingery, more or less curved, and somewhat soft and brittle coke commonly obtained from the coking coals highest in oxygen. Yet the cell structures of these cokes have certain common resemblances.

Both are characterized by small cells with thin cell walls (Figs. 16 to 19). The small size of the cells gives to these cokes a fine-grained appearance which sometimes leads to their being called dense. However, coke from low-volatile coal is of medium apparent specific gravity, whereas the coke

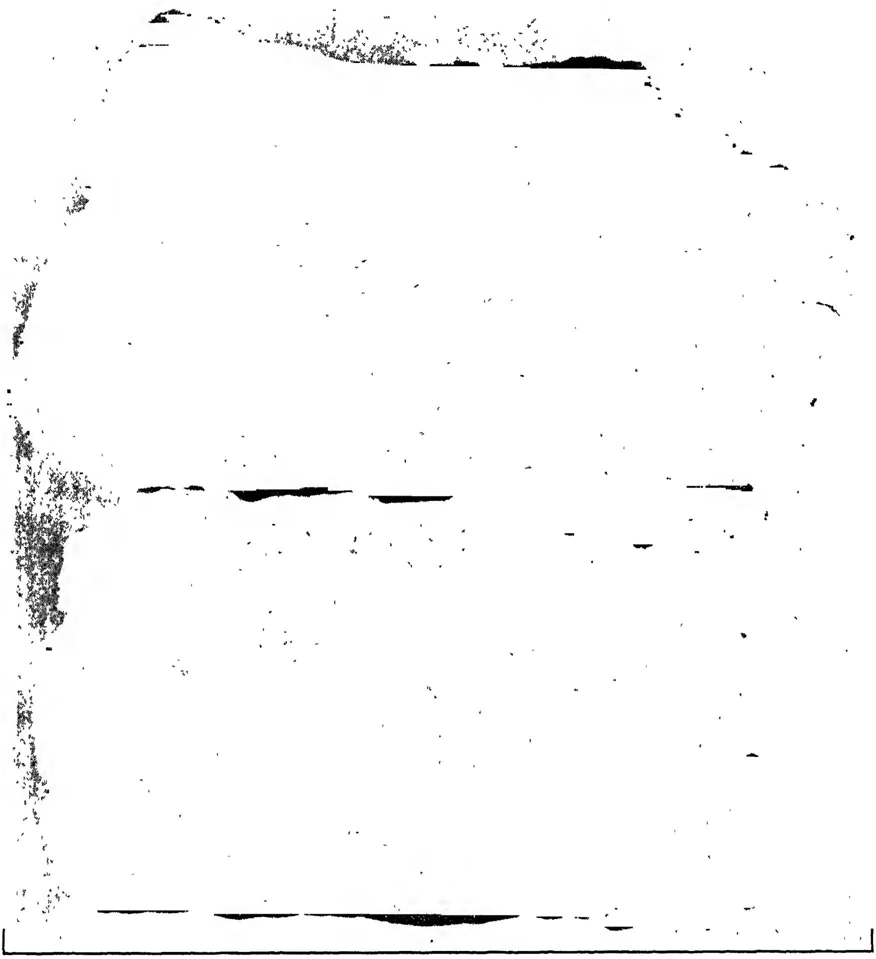

FIG. 18.—LONGITUDINAL SECTION OF COKE MADE IN BECKER TYPE OVENS FROM 35 PER CENT. VOLATILE MATTER, HIGH-OXYGEN COAL, NO. 6 SEAM, JEFFERSON COUNTY, ILL.

from high-oxygen coking coals will be found to have a very low apparent specific gravity, in fact it is the lightest in weight of all cokes.

The cell structure of the low-volatile coke is often extremely uniform, whereas the high-oxygen coals frequently produce a coke containing large cells irregularly distributed and areas of cells with thicker walls. Both

Fig. 19.—Longitudinal Section of By-product Coke from 34 Per Cent. Volatile Matter, High-oxygen Coal, No. 6 Seam, Franklin County, Ill. Cells Filled with White Composition. × 4.

Fig. 20.—Longitudinal Section of By-product Coke from 37 Per Cent. Volatile Matter, Slack Coal, Pittsburgh Seam, Monongalia County, W. Va. × 1.

Fig. 21.—Longitudinal Section of By-product Coke from 32 Per Cent. Volatile Matter Coal, Pittsburgh Seam, Westmoreland County, Pa. Cells Filled with White Composition. × 4.

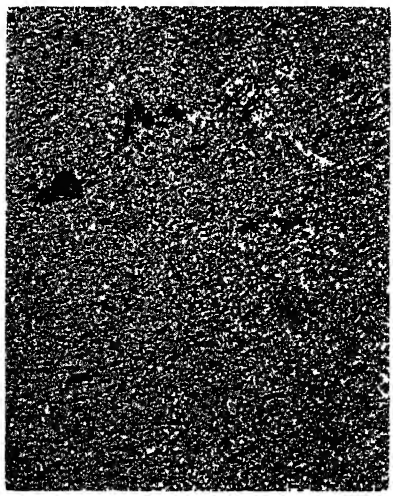

Fig. 22.—Longitudinal Section of By-product Coke from 26 Per Cent. Volatile Matter Coal, "B" Seam, Fayette County, Pa. × 1.

types of coal commonly contain more or less semicoking or non-coking material (other than slate). The above cokes have a cell size approximating that of Ramsburg and Sperr's Standard No. 1.[13] The method used by The Koppers Co. laboratories for sectioning coke has been described by Malleis.[14]

Certain high-volatile coking coals tend to produce a small-celled (No. ½ to 2)[14a] coke with very thick walls which is the hardest and densest of all types of coke (Figs. 20 and 21). Because of the large amount of shrinkage involved in the formation of a dense product from high-volatile

FIG. 23.—CELL STRUCTURE OF BY-PRODUCT COKE MADE FROM 25 PER CENT. VOLATILE MATTER, MIXTURE OF UPPER FREEPORT AND KITTANNING SEAM COALS, NORTHERN WEST VIRGINIA. CELLS FILLED WITH WHITE COMPOSITION. × 4.

FIG. 24.—LONGITUDINAL SECTION OF BY-PRODUCT COKE FROM 34 PER CENT. VOLATILE MATTER, GAS COAL, TAGGART SEAM, WISE COUNTY, VA. × 1.

coal, such coke is characterized by many longitudinal shrinkage cracks. If the coke pieces are large when formed, they will usually be found to consist of bundles of slender pieces which tend to break apart upon handling. The formation of coke of this type is not limited to coals having any particular ultimate analysis. Coals with a volatile content ranging from at least 28 to 37 per cent. may exhibit this behavior. The cause of the very dense structure appears to lie in the existence of certain physical conditions during the coking process.

[13] C. J. Ramsburg and F. W. Sperr, Jr.: By-product Coke and Coking Operations. *Jnl.* Franklin Inst. (1917), **183**, 391.

[14] O. O. Malleis: By-product Coke Cell Structure. *Ind. and Eng. Chem.* (1924), **16**, 901.

[14a] R. and S. standard.

Most of these coals respond very favorably to the addition of low-volatile coal, and the mixture will usually give a much more open cell structure and larger, stronger coke pieces. The addition of enough low-volatile coal may largely or entirely prevent formation of longitudinal shrinkage cracks.

Large-celled Coke

Medium-volatile coking coals characteristically produce a strong, blocky, silvery gray coke, of strikingly uniform and handsome appearance. The cells of those cokes from coals approaching the low-volatile

FIG. 25.—LONGITUDINAL SECTION OF 72-HOUR BEEHIVE COKE MADE FROM HIGH-VOLATILE COKING COAL, PITTSBURGH SEAM, FAYETTE COUNTY, PA. CELLS FILLED WITH WHITE COMPOSITION. × 25.

FIG. 26.—CARBONIZED "MINERAL CHARCOAL" (FUSAIN) CROSS SECTION IN POCAHONTAS 72-HOUR BEEHIVE COKE, MCDOWELL COUNTY, W. VA. CELLS FILLED WITH WHITE COMPOSITION. × 25. MINERAL CHARCOAL IS A NON-COKING CONSTITUENT.

class are apt to be small and thin-walled, whereas those from coals approaching the high-volatile group, although apt to be small, are likely to have rather heavy cell walls. However, many medium-volatile coals, particularly those with about 26 per cent. volatile content, give a coke with very large cells (No. 4),[14a] of uniform size and with profusely porous cell walls (Figs. 22 and 23). Such coke exhibits few cross fractures, but when broken, the fractured surface is smooth and at right angles to the length of the coke piece. The fractured surface facing toward the oven wall is noticeably brighter in appearance than the other side of the fracture. The flattened shape of the cells cause an almost flaky appearance.

Other coals, particularly those with a volatile content around 34 per cent., have also been observed to give coke with large uniform cells and porous cell walls, but the flaky silvery appearance of the fractured surface is lacking (Figs. 24 and 25).

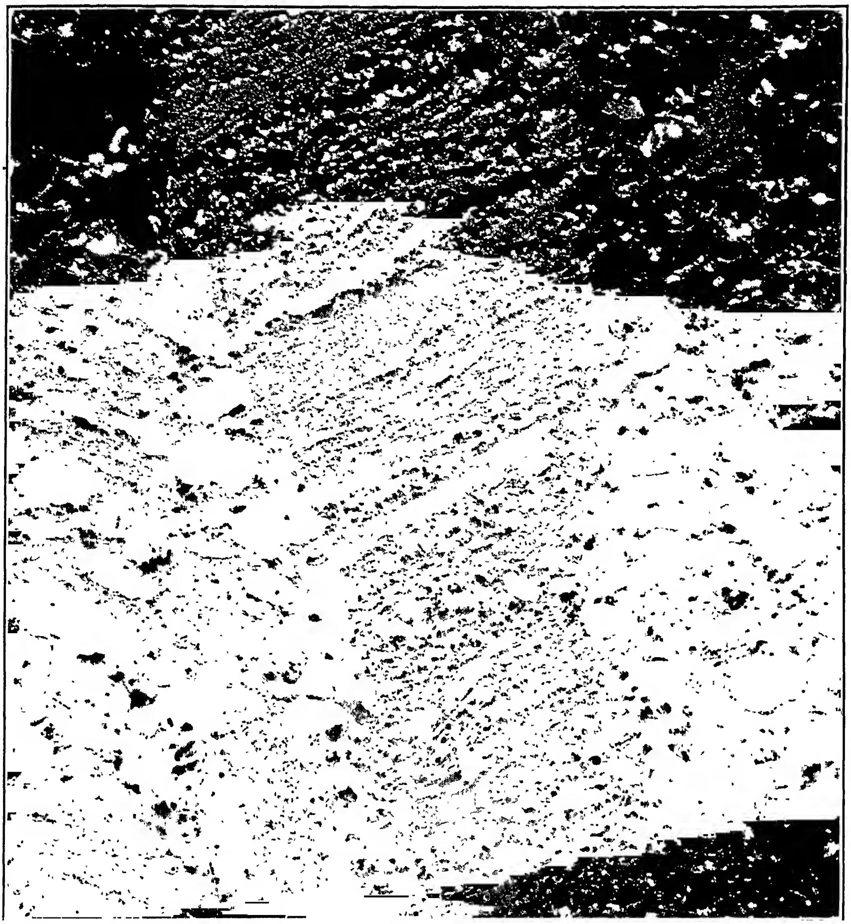

FIG. 27.—RESULT OF POOR PULVERIZATION OF A MIXTURE OF 80 PER CENT. PITTSBURGH SEAM COAL WITH 20 PER CENT. SOMERSET COUNTY, PA., LOW-VOLATILE COAL. THE CENTRAL AREA IS COKE FORMED FROM A FRAGMENT OF LOW-VOLATILE COAL SURROUNDED BY COKE FORMED CHIEFLY FROM THE HIGH-VOLATILE CONSTITUENT. × 18.

Coke with Cells of Medium or Irregular Size

Coals producing coke with cells of medium size may be considered as intermediate between certain of the coals mentioned above. Mixtures of dissimilar coals usually produce coke of this type.

Gas coals rather high in oxygen tend to produce coke with cells that average large but are irregular in size, wall thickness and general appearance. The individual constituents of banded coal usually possess different coking properties, and thus give rise to an irregular cell structure.

Low and high-volatile coals differ considerably in fusing temperature and physical behavior when heated. When such coals are to be coked together, a good degree of pulverization and thorough mixing is necessary to get the full value of the low-volatile material which is added (Fig. 27).

An Extended Range of Coals Can Now Be Commercially Coked

The most interesting and important fact brought out by the preceding discussion of coking coals, is that a very wide range of coal types are now available for the manufacture of by-product coke. Recent important advances in coke-oven design have greatly simplified the problem of coal selection. In most places a variety of coals acceptable for use in the modern ovens are available. Coal selection is a matter of choosing the coal or coals that appear to offer the most advantages. Even in isolated localities, where there is little or no choice among bituminous coals, a coke of satisfactory physical properties can usually be produced.

Yields and Value of Gas and By-products

The 49 million tons of coal carbonized in by-product coke ovens in 1924 gave the yields[15] shown in Table 2.

Table 2.—By-products Obtained from Coke-oven Operations in 1924

	Total Production	Average Yields per Net Ton Coal
Coke-oven gas		
Used at coke-ovens, M. cu. ft.	239,897,197	
Used in steel or affiliated plants, M. cu. ft.	187,171,883	
Distributed through city mains, M. cu. ft.	65,676,867	
Used under boilers, etc., M. cu. ft.	29,794,046	
Sold for industrial use, M. cu. ft.	18,561,057	
Total gas produced, M. cu. ft.	541,101,050	11,030
Tar, gal.	422,074,326	8.6
Ammonium sulfate (equivalent of all forms of ammonia produced), lb.	1,089,245,167	22.2
Crude light oil, gal.	128,956,955	[a]2.9

[a] Average for plants recovering light oil.

Table 3 gives the average annual value[16] of the products sold during a 3-year period.

[15] Production of By-product Coke and of By-products in 1924. U. S. Geol. Survey.

[16] U. S. Geol. Survey reports.

Table 3.—Average Value of Gas and By-products Sold during 1922, 1923, 1924

	1922	1923	1924
Coke-oven gas	Cents	Cents	Cents
Used in steel or affiliated plants, M. cu. ft	11.0	11.3	11.2
Distributed through city mains, M. cu. ft	35.7	35.5	35.8
Used under boilers, etc., M. cu. ft	6.5	5.4	5.8
Sold for industrial use, M. cu. ft	10.7	20.7	19.2
Tar, gal	4.0	4.4	4.6
Ammonium sulfate, lb	2.5	2.9	2.4
Crude light oil, gal	12.3	10.5	8.3
Motor benzol, gal	19 1	16.3	15.2

Coke comprises about two-thirds of the total products of coal carbonization, both in value and in weight. The most variable characteristic of the coals used is the coking property, and it is not surprising to find that coals for by-product coke ovens are selected primarily on the basis of the yields and quality of the coke, or coke and gas, which they will produce. By-product yields are also important, but, in general, this factor is of secondary importance in coal selection, except when it is necessary to choose between coals of similar coking properties.

It is impossible to generalize satisfactorily on the by-product yields obtainable from coal, because of the considerable range in chemical composition of the coals of each class discussed and because by-product yields also depend on operating conditions. Furthermore, coals even from the same mining district may possess marked individualities.

The by-product yields obtainable from many coals and coal mixtures under particular operating conditions, have been established by long use, but for unknown coals, special methods must be used to determine the yields. Separate by-product recovery apparatus has sometimes been attached to single test ovens located in the regular oven batteries of coke plants, but this method has not met with much favor as a practicable and reliable test.

In the writer's opinion,[17] the most satisfactory method is the distillation of coal in the laboratory, under conditions which duplicate actual coke-oven conditions as far as possible; the by-products are collected, and the yields determined. It is rapid, reliable, and requires but a small sample of coal. Long experience has proved that the success of this empirical test depends upon the rigid control of test conditions by chemists experienced in this work and frequent comparison of laboratory results with actual plant yields for the same coals.

[17] H. J. Rose: A New Electric Furnace for the Determination of the By-product Yields of Coal. (Unpublished.) Presented before the Gas and Fuel Section of the Amer. Chem. Soc., Pittsburgh, Pa., Sept. 7, 1922.

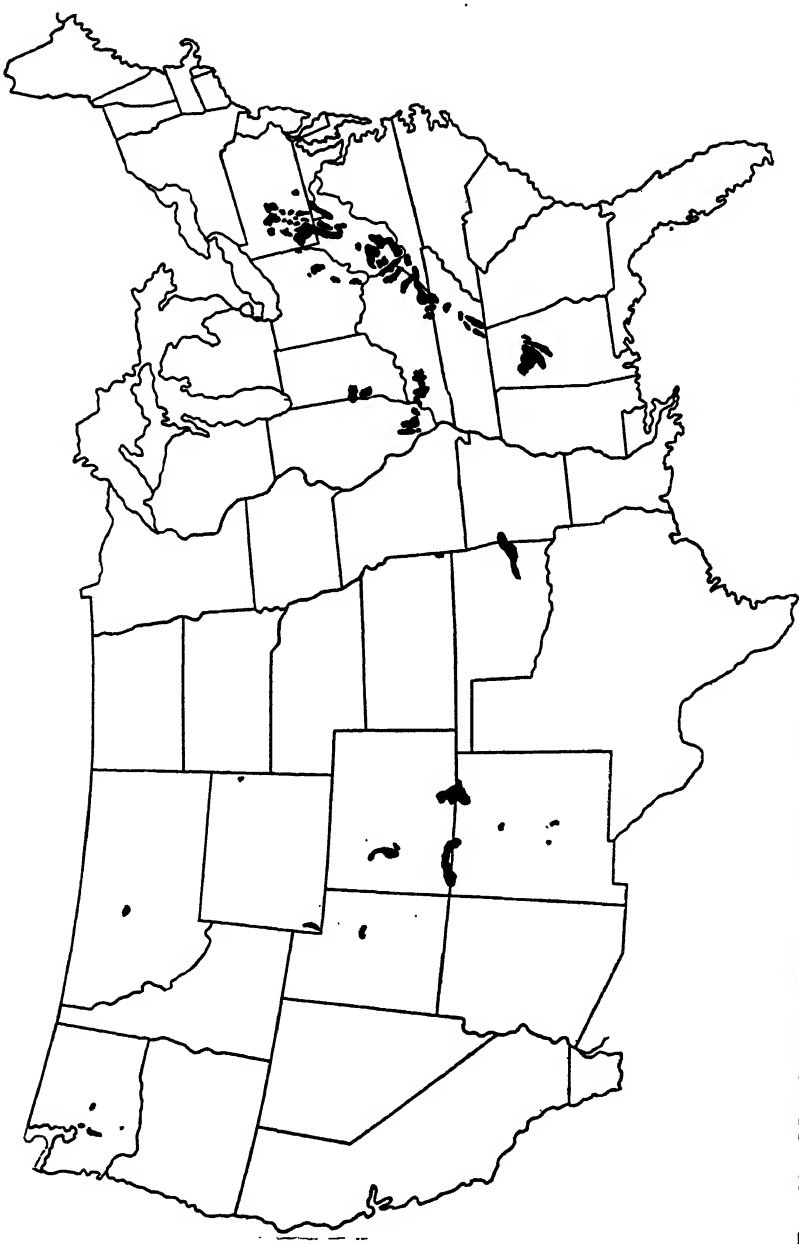

FIG. 28.—MAP SHOWING MINING DISTRICTS OR COAL AREAS OF THE UNITED STATES, WHICH CONTAIN COKING COAL.

Coking Coal Deposits in the United States

Fig. 28 shows the coal-mining districts or coal areas of the United States which contain coking coal. The shaded areas east of the Mississippi River represent districts which can produce coking coal of reasonable purity (in some places the coal must be washed). Known areas of coking coal west of the Mississippi River, are smaller and more widely dispersed, and the shaded areas indicate, in general, the extent of those coal deposits which are of semi-bituminous or bituminous rank and are said to contain coal having coking properties.

At present there are but three by-product coke plants in the western half of the United States. They are in Colorado, Utah and Washington, and use coal produced in those states respectively.

The 41 million tons of coal used in the manufacture of by-product coke in 1922, were produced in the following States:[18]

Table 4.—*Source of Coal Used in By-product Coke Ovens in* 1922

State in Which Coal was Mined	Net Tons Produced
Pennsylvania	14,243,751
West Virginia	11,489,232
Alabama	5,114,121
Kentucky	7,185,427
Illinois	840,132
Virginia	780,926
Colorado	528,956
Indiana	466,235
Tennessee	37,061
Oklahoma	26,006
Washington	9,390
Arkansas	Small amount
Imported	369,290
Estimated total	41,091,127

About 15 to 20 per cent. of all the coal used in by-product coke ovens in the United States, and most of the coal so used in Alabama, Colorado, Tennessee and Washington, is washed coal.

Effect of Coal Mixing, Coal Preparation, and Oven Operating Conditions on Coke Quality

The size and strength of coke can be modified and controlled to a very considerable extent by mixing two or more coals together; by varying the preparation and condition of the coal as charged; and by varying oven-

[18] F. G. Tryon: Coke and By-products in 1922. U. S. Geol. Survey, Min. Resources of the U. S., 1922, Part II.

operating conditions. A large amount of data on these subjects has been accumulated, but even a brief discussion of their possibilities is hardly practical in this paper.

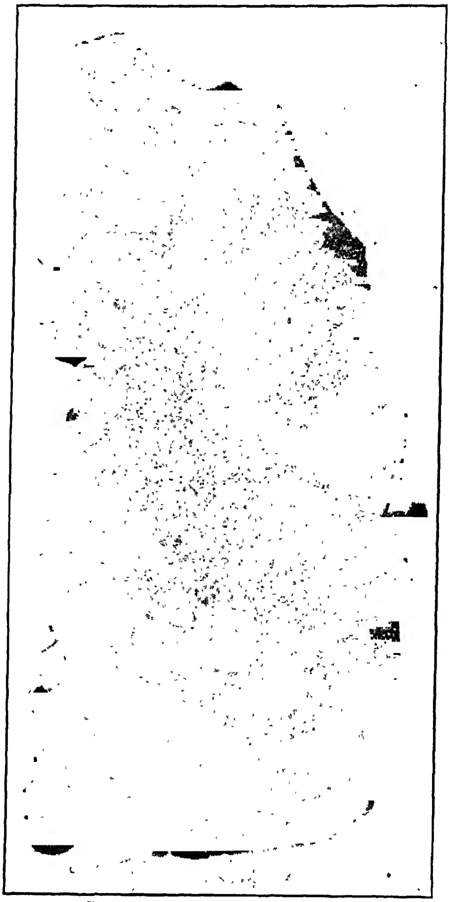

FIG. 29.—LONGITUDINAL SECTION OF BOX-TEST COKE MADE FROM 32 PER CENT. VOLATILE MATTER AND 8.8 PER CENT. ASH "SPLINT COAL," OCCURRING AS A NARROW BAND IN A W. VA. HIGH-VOLATILE COKING COAL. THE ULTIMATE ANALYSIS OF THE SPLINT COAL IS SIMILAR TO THAT OF THE BRIGHT COAL IN THE SAME SEAM. (SEE FIG. 30.)

Coal preparation and condition includes such factors as the degree of pulverization and range of sizes; the moisture content of coal; the quantity and size of pieces of shale, slate, or other mineral matter present; the amount and nature of poorly coking or non-coking material (Figs. 26, 29 and 30); weathering or oxidation of the coal. Most of these variables exert several different and distinct influences, so that the correct

application of the various principles which have been evolved, requires considerable skill and practical experience.

The more important oven-operating conditions affecting coke size and strength are oven temperature, degree of under-coking or over-coking,

FIG. 30.—LONGITUDINAL SECTION OF BOX-TEST COKE MADE FROM 36 PER CENT. VOLATILE MATTER AND 2.3 PER CENT. ASH "BRIGHT COAL," FROM SAME SEAM IN WHICH SPLINT COAL (SEE FIG. 29) OCCURRED.

uniformity of oven heating and oven width. The coking time depends on all of these variables.

CHEMICAL AND PHYSICAL CHARACTERISTICS DESIRED IN COKE

Blast-furnace Coke

Ash.—The permissible ash content of blast furnace coke in each locality depends upon the purity of the available coals and will range from

about 8 to 16 per cent. The best cokes contain less than 10 per cent. of ash, and a strong prejudice exists against coke having more than about 12 per cent., if better coke is available. This means that the coals used should contain not over 9 per cent. ash, which is the standard specification of the American Society for Testing Materials[19] for both gas and coking coals.

In the Pittsburgh district, large quantities of crushed run-of-mine Pittsburgh seam coal having as much as 10 per cent. ash, are coked in by-product ovens. When such coal is charged coarsely crushed, a considerable proportion of the shale and slate passes into the nut and breeze coke sizes, and the blast-furnace coke may contain only 12 or 13 per cent. ash. If the coal is pulverized, the ash is more uniformly distributed throughout the various coke sizes.

Even after washing, the coking coal available in some localities produces a coke having as high as 16 per cent. ash. But it must not be forgotten that a blast-furnace plant using a high-ash coke with a very pure or limy ore, may be in just as favorable a position as a plant using a very pure coke with a low-grade ore.

Calculations that have been made from time to time, show that each additional per cent. of ash in coking coal reduces its value to the blast-furnace plant about 30 c. per ton on an average, and operators are unquestionably justified in demanding the lowest possible ash in coke. Shale and slate have an injurious effect on coke size and strength, the full extent of which is probably not generally realized.

Sulfur.—The A. S. T. M. specifications previously quoted state that coking coal shall be of such composition that the coke produced shall contain not more than 1.3 per cent. sulfur, for blast furnace coke. Less than 1 per cent. sulfur is preferable, and coke having as much as 1.50 to 1.75 per cent. sulfur is not considered acceptable at many furnaces.

The percentage of sulfur eliminated during the coking process depends upon the coals used, but will average around 40 per cent. for coals giving a 70 to 75 per cent. coke yield. In a general way this means that the percentage of sulfur in coke is from 0.1 to 0.3 less than in the coal from which it was made, but the sulfur content of coke is sometimes as high, or even higher, than the coal used.

It is a fortunate dispensation that many of the coking coals in the far west that are high in ash even after washing, are very low in sulfur.

Phosphorus.—The iron used in both the acid open hearth and Bessemer processes must be very low in phosphorus, and coke used in making such iron should preferably contain less than 0.010 per cent. of this element. All of the phosphorus in coal passes into the coke, and all of the phosphorus in the materials charged into a blast furnace passes

[19] A. S. T. M. Standard Specifications for Gas and Coking Coals. Serial Designation, D 166-24.

into the iron. A coal containing as much as 0.020 per cent. phosphorus would generally be considered of limited value, as most American blast-furnace cokes are very low in phosphorus.

The phosphorus content of coke used in making basic iron is of little importance, as the basic steel-making processes readily remove this impurity

Foundry iron is usually required to contain a considerable amount of phosphorus and high-phosphorus ore is commonly used, or phosphatic rock may be added. The phosphorus content of the coke is usually of not much importance.

Size and strength.—Present American practice requires the use of well-sized coke; lumps larger than 4 in. are usually crushed. Coke breeze (fines smaller than about ¾ in.) is detrimental to furnace operation and should always be removed as thoroughly as possible. The nut or domestic size (about 1¼ to ¾ in.) may be removed if it can be disposed of to advantage, or is often screened out and charged separately into the blast furnace.

The total percentage of blast-furnace coke remaining on a 2-in. square mesh screen is often used as the index of size, and most blast-furnace operators desire the largest possible percentage of coke between the 2-in. and 4-in. sizes.

The only reliable way to determine the size and strength of the coke that can be produced from a given coal or coal mixture, and the yields of furnace, nut and breeze sizes, is to make full-oven coking tests under the desired operating conditions

Combustibility—There has been an active and extensive discussion in recent years as to the existence and importance of differences in coke combustibility. The subject is too complex for discussion here, but a few comments are required. The term "coke combustibility" has no accepted definition, but probably the best usage is that which refers to the *chemical reactivity of coke towards carbon dioxide* or oxygen at high temperatures.

Following Brassert's[20] original discussion of this characteristic of coke, most blast-furnace operators seem to have accepted his ideas at least in part, and some experienced men are thoroughly convinced of the soundness of the theory. The chief evidence presented against this theory consists of the papers of Perrott and Kinney,[21] and Sherman and Blizard.[22] After a very extended graphic study of the data presented

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[20] See J. E. Johnson, Jr.: Principles, Operation and Products of the Blast-furnace. (1918), 171.

[21] G. St. J. Perrott and S. P. Kinney: Combustion of Coke in Blast-furnace Hearth. *Trans.* (1923), **69**, 543.

[22] R. A. Sherman and J. Blizard: Combustion of Blast-furnace Cokes in Fuel Beds. *Trans.* (1923), **69**, 526.

in these papers, the writer stated[23] that the conclusions reached were not justified by the data presented. A later paper by one of the same authors[24] confirms this opinion, as the later work showed that larger differences did exist in the chemical reactivity of different cokes in fuel beds.

All of the by-product cokes which were tested were found to be far more reactive than any of the beehive cokes tested. This discovery is interesting in view of the now well established fact that by-product cokes give distinctly better efficiencies in the blast furnace than beehive cokes. In any practical study of coke combustibility, the effect of coke size must not be overlooked

Density and structure.—Most blast-furnace operators prefer a coke of medium density with fairly large cells and thin or porous cell walls, but there are many variables in furnace operation to confuse individual issues, and a unanimity of opinion on this point can hardly be expected. At some plants a strong prejudice exists against a heavy coke, that is one that gives a high skip weight.

Uniformity.—Uniformity of coke is vital to successful and economical furnace operation. Uniform ash and sulfur content are important in maintaining the desired slag composition. Uniformity of coke size is desired, to help prevent the segregation of sizes when the coke is dropped into the top of the furnace. Coke is usually charged by volume instead of by weight, and variation in the apparent specific gravity of dry coke or the size of the coke, may seriously affect the ratio of coke to ore. For these and other reasons it is evident that a coke of very uniform quality may be more desirable than one of better average quality but lacking in uniformity.

Foundry Coke

Foundry coke is used for metal melting, and the chief requisites are purity, large size and good strength, and a minimum reactivity with carbon dioxide gas. As the function of foundry coke is simply to furnish heat by combustion, carbon monoxide gas at the top of the cupola represents wasted energy. This differs from the conditions in the blast furnace, where the complete reduction of carbon dioxide to carbon monoxide in the hearth and bosh is essential to the reduction of iron ore higher in the furnace.

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[23] H. J. Rose: Coke Structure and Combustibility. (Unpublished.) Presented before the Eastern States Blast-furnace and Coke Oven Assn., Oct. 5, 1923, Pittsburgh, Pa.

[24] R. A. Sherman: Combustibility of Blast-furnace Coke. *Iron Age* (1925) 1043.

Specification Limits for Chemical Composition

The following specification limits[25] for the chemical composition of foundry coke have been adopted by the American Society for Testing Materials:

	Percentage in the Dried Sample
Volatile matter	Not over 2.0
Fixed carbon	Not under 86.0
Ash	Not over 12.0
Sulfur	Not over 1.0

The specifications describe car and laboratory sampling, and provide for rejection.

A high-sulfur content is very objectionable and many private specifications require less than 1 per cent. of this element. The phosphorus content is usually unimportant. A low-ash content is, of course, desirable. The coke should be thoroughly carbonized.

By-product foundry coke is usually screened over a 3-in. grizzly. Foundrymen almost universally require a large, hard, strong coke and are accustomed to pay a premium in order to obtain coke which meets their standards of size, strength and purity. In order to obtain a good yield of foundry-size coke, one or more of the following procedures may be resorted to:

1. A large percentage (30 per cent. or more) of low-volatile coal may be mixed with a selected high-volatile coal. The proper coals and proportions are determined by experiment. The volatile matter of the coal mixture is usually 25 to 28 per cent. This is a common method for producing foundry coke.
2. A medium volatile coal or mixture of such coals may be used. This is satisfactory where the proper coals can be obtained.
3. By the use of low temperatures and a long coking time, the size of the coke pieces can be much increased.

Domestic Coke

The most important characteristics that depend on the coal used, are the ash content, fusing temperature of ash, and density of the coke.

The ash content of cokes sold for domestic use varies widely from plant to plant, depending upon the kinds of coal which are available, and the purposes for which the coke plants are being operated. In some cases coals containing less than 6 per cent. ash are regularly used, and the 8 or 9 per cent. ash coke which is produced is suitable in this respect for the most strongly competitive markets. Domestic cokes containing 10 to 15 per cent. ash may be classed in most localities as good to

[25] A. S. T. M. Standard Specifications for Foundry Coke. Special Designation D 17-16.

fair, respectively. Nut-size coke of still higher ash content is produced by some plants, but it is not made with the requirements of the domestic trade in view, and is usually sold at a considerably lower price than first quality domestic coke.

A fusing temperature of 2500° F. or over is desirable for ash, and the higher this temperature the better. However, the permissible fusing temperature depends to a considerable extent upon the amount of ash in the coke, the coke size and the nature of the domestic use to which it will be put. Certain low-ash cokes having an ash fusing temperature of 2300° to 2400° F. have acquired an excellent reputation. The American Society for Testing Materials standard specifications for gas and coking coals (previously mentioned) give 2200° F. as the minimum, but this is unquestionably a low limit.

Since the cubic foot weight of coke is lower than that of coal, a dense coke is usually desired in order that a sufficient weight can be charged into domestic appliances at each firing.

The various domestic coke sizes should be closely screened, and the proper size selected for each appliance. Domestic coke should therefore be hard and tough to prevent breakage and the formation of fines on handling.

Water-gas Coke

The most serious operating problems connected with water-gas manufacture are due to the presence of clinker and its removal. The amount of ash present in coke has an important bearing on the frequency and cost of clinker removal, and the output of water gas. It is hard to overestimate the advantages of low-ash coke, both from the standpoint of ease of operation and cost of gas made. Coals containing 6 per cent. or less ash are indicated for the manufacture of high-grade water-gas coke.

The ash fusing temperature is, of course, important, and in general should range from 2400° to 2500° F. with 2300° F. as a lower limit. Many operators would not put any upper limit on the ash fusing temperature, but some experienced men state that an ash of too high a fusing temperature causes troublesome wall clinker. The ash fusing temperature that is best for a particular plant will depend somewhat upon operating conditions.

A low-sulfur content is desirable, as part of the sulfur passes into water gas as hydrogen sulfide.

Water-gas coke preferably consists of lumps ranging from 2 to 4 in., but much of the coke used for this purpose has had only the fines removed. Breeze is objectionable, and the coke should be strong enough to permit handling without excessive breakage. Uniformity in the size of coke as charged is greatly to be desired.

DISCUSSION

R. H. Sweetser, Columbus, O.—In the work Mr. Rose has done on the coal that goes into the coke oven, it is possible to calculate the yield of total coke from the analysis of the coals?

H. J. Rose.—The coke yields obtained in regular plant operation are related, but not directly proportional, to the coke yields indicated by the ordinary volatile-matter figure as reported in the proximate analysis of coal. The discrepancy between actual and laboratory coke yields ranges from about 2 to 6 per cent. and varies with the volatile-matter content of the coal. Our own practice is to estimate practical coke yields from the results of special laboratory tests, and we think that in this way coke yields can be predicted as accurately as the coke plant operator is usually able to measure them.

R. H. Sweetser.—Suppose there was 30 per cent. volatile in the coal?

H. J. Rose.—The yield of total dry coke would probably be about 74 per cent. of the dry weight of the coal charged.

R. H. Sweetser.—Then the chemist reports 30 per cent. of volatile matter in a coal when there is really not that amount present?

H. J. Rose.—The method and rate of heating a coal affect the amount of volatile matter which passes off. In the case of the coal just mentioned, 30 per cent. of volatile matter would be liberated if a 1-gram sample was rapidly heated in a platinum crucible, but only about 26 per cent. of volatile matter would pass off if the coal was coked under coke oven conditions. In the latter case, the greater degree of cracking of the volatile constituents, would result in a decreased weight of volatile matter and an increased weight of coke.

T. M. Chance, Philadelphia, Pa.—Dorrance, while investigating various briquetting processes, found that often the refuse was higher in volatile than the pure coal, because the volatile, combustible matter, reported by the laboratory on an empirical basis, contained water of combination in the silicates from the refuse.

Dr. St. John, has any attempt been made to isolate the vitrain and similar compounds in coking coals by the use of an X-ray apparatus?

A. St. John, New York, N. Y.—I am unable to answer that question. With respect to the greater amount of fixed carbon appearing in the coke as manufactured as against that shown by the laboratory tests, that is something that is to be expected. The hydrocarbons, which constitute

the volatile matter, are principally carbon and very little hydrogen, and there is a possibility of recovering a considerable amount of carbon from the volatile matter, if you conduct your process with a sufficient degree of slowness of heat. That, of course, is the basis of the binder action in the manufacture of artificial carbon products, such as carbon electrodes and brushes.

H. C. PORTER, Philadelphia, Pa. (written discussion).—This paper excellently presents practically all that can be said as to methods that have proved useful in actual practice for judging coals for coke manufacture. The cuts of coke sections are particularly instructive.

In the discussion of what happens to the coal when it is gradually heated to form coke, the statement is made that a coal has a definite fusing temperature. I think an exact or closely defined temperature cannot be meant—one that differs with different kinds of coal. Although different kinds of coal show indications of beginning to soften at different temperatures, the point for each is not well defined as for a homogeneous simple substance.

Also, in this connection, it is said that decomposition begins at the fusion point or slightly above it. Such a characterization of coking coals has been advanced before, but I believe that it is extremely possible that all coals undergo some decomposition at temperatures lower than their softening points, and that this is not the true explanation of the coking characteristic or the lack of it. Possibly the author does not intend to give that impression.

In the treatment of the connection of the ultimate composition of coals with their coking adaptabilities, the reader is left somewhat uncertain as to how the chart of plotted results of ultimate analyses (Fig. 8) can be used to advantage in selecting the best coals for coking. What are the limits referred to, of area or segment on the diagram where "coking properties and by-product yields may be predicted with reasonable accuracy, provided . . . "? Apparently the coking-coal area shown in Fig. 8 covers largely the whole area of semi-bituminous and bituminous coals in Fig. 4.

In the discussion of high-oxygen coals and their use in coke ovens, I am interested to note that he says higher temperatures of coking are required for best results for these coals than for the lower-oxygen coals. It has been my impression that, although these coals are well understood to require more heat for their carbonization, possibly because of their forming more CO_2 and water of decomposition, the best treatment to give them, considering both coke quality and by-products, is in a narrow oven without too much increase of temperature.

H. J. ROSE.—The fusing or softening temperature of a coking coal is of course not so definite as the melting point of a pure chemical compound.

However, the softening temperature as obtained by published methods is quite sharply defined, and can be checked to within several degrees.

The relation between the fusing temperature of coal, and the temperature at which gases of decomposition begin to appear in appreciable and increasing amounts, varies with individual coal samples. A detailed study of a large number of American and foreign coals warrants the empirical statement that the two temperatures are usually similar in the case of good coking coals. The decomposition temperature of coal varies in a general way with its rank. Coal is a very complex organic material, and it is reasonable to consider that it undergoes some change at temperatures below which gases of decomposition are evolved.

Fig. 8 shows the approximate ultimate analysis limits of coking coals and divides them into five areas to which familiar names are applied. Each of these five groups is defined, and their typical coking behavior discussed in as detailed a way as is practical for a whole group. It is not possible to include on a single graph of the size of Fig. 8, all of the data which are necessary to permit the detailed prediction of coking properties and by-product yields. For this purpose it would be necessary to prepare a separate chart for each of the variables under consideration, showing "iso" lines for that variable. Another method used by the writer, is a very large wall chart, which affords sufficient space to write down data beside each point representing the ultimate analysis of a coal of known coking behavior and by-product yields. The probable behavior of an unknown coal is then estimated from its location with respect to known coals.

It has been the writer's observation that high-oxygen coals produce the best coke when carbonized in narrow ovens having a high and very uniform temperature, thus giving a rapid rate of heat input. Owing to the greater B. t. u. requirement for coking high-oxygen coals, the use of a higher coking temperature is not accompanied by a short coking time. In fact, as long or even somewhat longer coking time than would be necessary for good coking coals at normal heats, may be used in actual practice. This has led to some confusion since a longer coking time generally (*though not in this instance*) indicates a lower oven temperature.

X-ray Studies of Coal and Coke

By Ancel St. John,* Ph. D., New York, N. Y.

(Pittsburgh Meeting, October, 1926)

I. PRELIMINARY SURVEY

Introduction

During a session on coal and coke at the February, 1926, meeting of the American Institute of Mining and Metallurgical Engineers, the writer called attention to the important work on the X-ray analysis of coal[1,2] being carried on in England by C. Norman Kemp and his associates. This aroused so much interest that a paper on the subject was requested for the October meeting. It has seemed advisable to make this paper a preliminary survey of the present status in the application of X-ray methods to coal problems and to follow it from time to time with further articles as the art develops.

A survey of this sort naturally divides itself into two parts, radioscopic analysis of the distribution of constituents through the body of the coal, and diffraction analysis of the state of association of the elementary components. In this article the subject matter concerning radioscopic analysis has been drawn almost entirely from the writings of Kemp and his associates and the illustrations have been specially furnished by him. It is a pleasure to record here a great indebtedness for this courtesy and cooperation. The subject matter and illustrations concerning diffraction analysis are taken from the author's unpublished researches.

Nature and Properties of X-rays

Before proceeding it is well to recall the nature and some of the properties of X-rays, as well as our present conception of atomic structure. According to the accepted theory atoms are composed of electrically-charged particles. A positively-charged center, called the *nucleus*,

* Consulting physicist.

[1] C. Norman Kemp: The X-ray Analysis of Coal with Scientific and Technical Applications. *Trans.* Inst. Min. Eng. (Feb., 1924).

[2] Wm. McLaren: The Scientific Control of Coal Washing by the Combined Application of Ash-characteristic Curves and X-ray Examination. *Trans.* Inst. Min. Eng. (June, 1925).

is surrounded by a number of negatively-charged particles, called *electrons.* These are believed to move in orbits forming a sort of miniature solar system about the nucleus. The mass and charge of an electron is the same no matter with what atom it may be associated. The mass and charge of the nucleus and the number of associated electrons are proportional to the atomic number of the element. In addition to their normal motions about the nucleus the electrons may vibrate in other ways if the atom is agitated sufficiently, and in so vibrating may set up electro-magnetic or "light" waves. The force between the nucleus and an electron decreases with increasing distance between them but increases with increasing charge on the nucleus. It may be shown that as this force increases the work required to agitate an electron and the frequency of the resulting vibration both increase, very much as a short stiff spring requires more energy to set it in motion and vibrates more rapidly than a long, flexible one.

Visible light is produced when agitation of the atom displaces one of the electrons forming the outer portion of the atom; X-rays arise when the inner electrons are displaced. This is usually accomplished by the impact of a free electron moving with speed enough to penetrate to the interior of the atom. The X-rays given off in this manner have wave-lengths peculiar to the atom and are called the characteristic X-rays of the element. In addition the stoppage of the free electron also sets up a series of waves known as the general or independent X-rays, of various wave-lengths exceeding a minimum determined by the speed of the impinging electrons.

Production of X-rays

In this country the X-rays used for technical purposes are usually produced in the Coolidge type of X-ray tube. This consists of a highly-evacuated, thin-walled glass bulb containing a spiral tungsten filament, sometimes called the cathode, and a heavy metal block, usually tungsten or molybdenum, sometimes called the target or anode. The supply of free electrons is obtained by heating the filament with an auxiliary current, as in a radio tube. The speed is imparted to them by impressing the voltage of a high tension transformer between the filament and target, and using suitable means, such as a rotating high tension switch, a rectifying tube, or the rectifying action of the X-ray tube itself when the target is kept cool, to insure the flow of electrons only from filament to target.

Absorption of X-rays

X-rays are only slightly visible to the human eye and produce very serious physiological effects when they fall on any part of the body frequently or for a long time, hence some other method than direct

vision must be used for detecting them. When they traverse a substance part of their energy is absorbed, part is scattered and the balance passes through. Some of the absorbed energy may be given off again as longer X-rays, known as secondary rays, or sometimes as visible light or fluorescence. In some cases part of the absorbed energy may cause physical or chemical transformations, such as ionizing a gas or activating a photographic emulsion. The intensity of such effects depends upon the intensity of the rays producing them and may be used as means of detecting or recording the radiation.

The amount of the absorption and scattering depends upon the wavelength of the X-rays, upon the number and arrangement of the electrons in the absorbing atoms, and upon the number of atoms of each kind which are encountered, but is practically independent of the way the atoms are arranged. Expressed in another way, it depends somewhat upon the atomic weight of the target but principally upon the voltage across the tube and the composition, density and thickness of the absorber. In general, for a given voltage the absorption increases rapidly with the atomic weight of the absorber; for a given absorber it decreases rapidly with the voltage. When an appropriate voltage is used it is possible to determine the distribution of abnormal material, such as ash in coal, either by observing a fluorescent screen in a darkened room or by a photographic record. The latter is more sensitive and reliable.

Historical Summary of Radioscopic Work on Coal

Soon after the discovery of X-rays in 1895 by Roentgen[3] Prof. H. Couriot[4] published an important article laying the foundation for all subsequent work on radioscopic analysis of coal and coke. In 1899, J. Daniel[5] described attempts to determine the ash content by radioscopic methods. A paper by Garrett and Burton[6] in 1912, and the discussion thereof by a number of eminent authorities showed that X-ray examination could supply valuable information as to the distribution and mode of aggregation of ash in coal and as to the origin and structure of coal and coal seams. Later Iwasaki[7] used radioscopic methods in selecting samples of coal to be submitted to various tests and made a series of

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[3] W. K. von Roentgen: Ueber eine neue Art von Strahlen. Sitz. d. Wurz. Phy. med. Gesell. (1895).

[4] H. Couriot: Examen et Analyse des Combustibles Mineraux. *Bull.* Soc. Ind. Min. (1898).

[5] J. Daniel: Application des Rayons de Roentgen a l'examen des Combustibles Mineraux. *Ann.* Mines Belgique. (1899).

[6] Garrett and Burton: The Use of X-rays in the Examination of Coal. *Trans.* Inst. Min. Eng. (1911–12).

[7] C. Iwasaki: A Fundamental Study of Japanese Coal. *Tech. Rpts.* of Tohoku Imperial Univ. (1920–22).

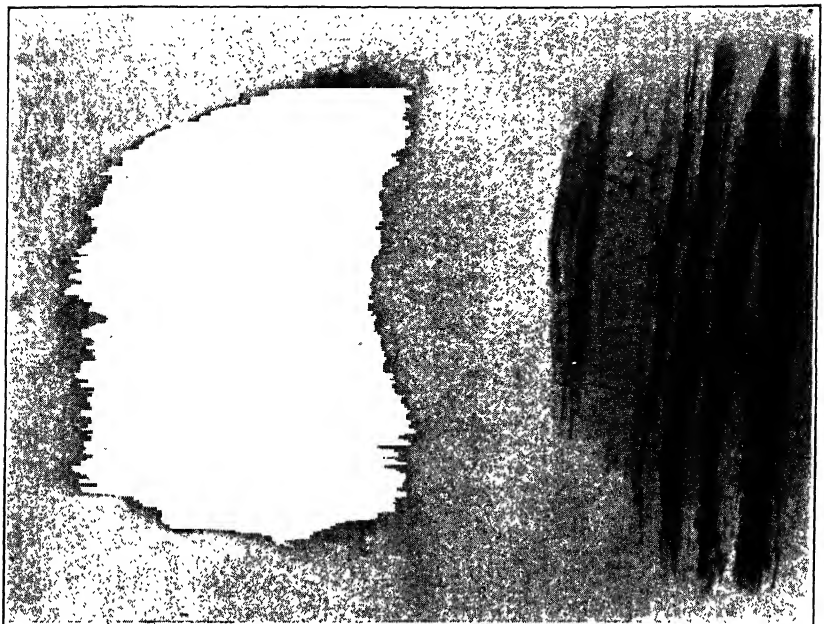

FIG. 2.—RADIOGRAPH OF ANTHRACITE BORE CORES.

FIG. 1.—RADIOGRAPH OF A LUMP OF COAL.

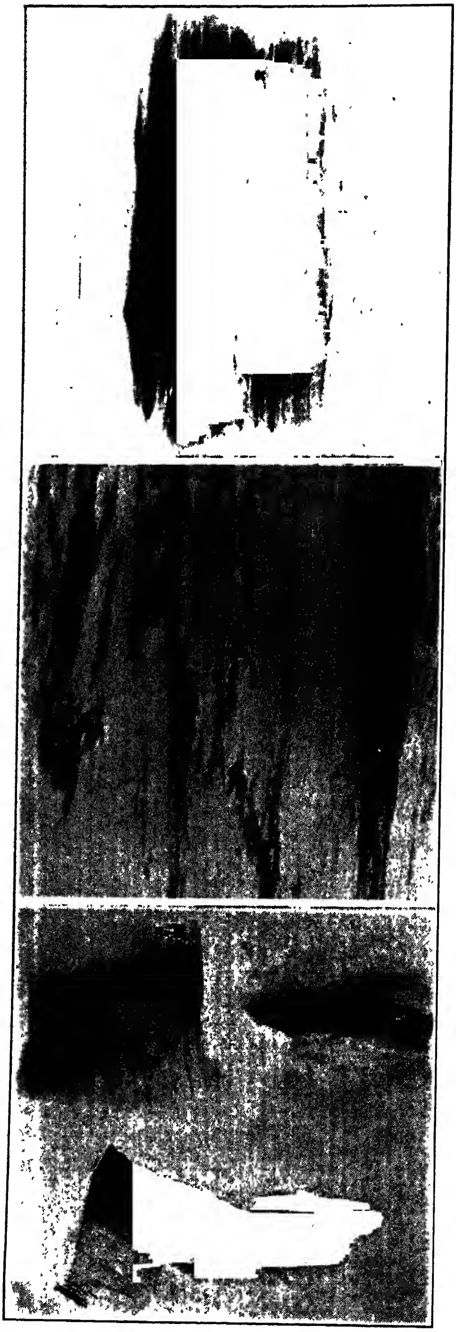

FIG. 3.—RADIOGRAPH OF PIECES OF BITUMINOUS COAL.

FIG. 4.—RADIOGRAPH OF A PIECE OF BITUMINOUS COAL.

FIG. 5.—RADIOGRAPH OF A FRAGMENT OF ANTHRACITE COAL.

radiographs of thin sections to supplement microscopic analysis. In 1923 Professor Henry Briggs[8] presented radiographs of Welsh anthracite prepared by Kemp and described a method for isolating ash material from carbonaceous matter so as to determine their respective ash contents. Finally Kemp and his associates, in a series of papers beginning in 1924,[9, 10] described methods and equipment for making rapid and reliable determinations of the distribution of ash materials in the mass of coal or coke, and for evaluating the ash in a sample.

Method of Radioscopic Analysis

The radioscopic analysis of coal and coke is based upon the variation of absorbing power with composition. Coal is essentially a mixture of combustible organic constituents made up of finely-divided free carbon, hydrocarbons and substitution complexes plus non-combustible mineral

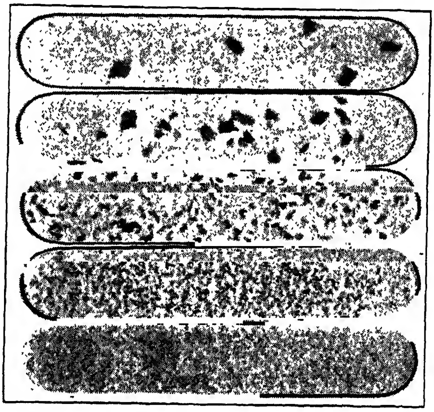

Fig. 6.—Radiograph of graded samples of coal, showing distribution of non-combustible material.

matter such as calcite or pyrites. Carbon, hydrogen and oxygen are all very transparent to X-rays. Sulfur and calcium are much less transparent and iron is relatively opaque to the wave-lengths used. Hence in a radiograph of a lump of coal (Fig. 1) the light areas indicate relatively pure carbonaceous material, darker areas indicate the presence of some sulfur, calcium, iron or other elements of higher atomic weight than carbon, and the very black portions are dominantly of high atomic weight.

The possibilities of radioscopic analysis are at once evident. During the geological exploration of a coal seam radiographs of the borings, such as the anthracite bore cores shown in Fig. 2, will give valuable

[8] H. Briggs: The Anthracite Problem. *Proc.* South Wales Inst. (1923).

[9] C. Norman Kemp: *Op. cit.*

[10] Wm. McLaren: *Op. cit.*

information as to the character, amount and distribution of combustible and non-combustible constituents. So also will radiographs of lumps, such as the pieces of bituminous coal illustrated in Figs. 3 and 4, or the fragment of anthracite in Fig. 5. During the preparation of the coal for market further information as to the distribution of the non-combustible material in the different sizes can be obtained from pictures of graded samples (Fig. 6), and, similarly, the distribution of ash in coke is thoroughly disclosed (Fig. 7).

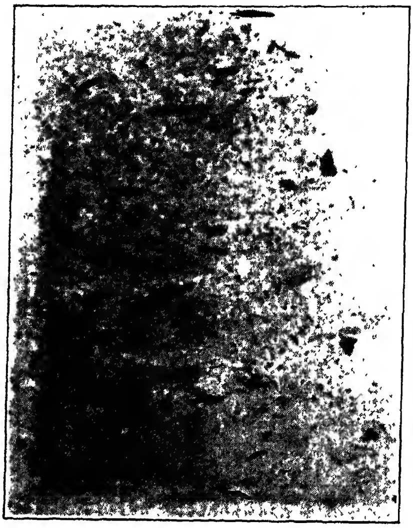

FIG. 7.—RADIOGRAPH OF COKE SHOWING DISTRIBUTION OF ASH.

RADIOSCOPIC CONTROL OF COAL-WASHING

The method is particularly valuable in the analysis and control of coal-washing processes. Kemp[11] and McLaren[12] have directed special attention to this and have developed the technique and equipment to a high degree of efficiency. The method consists essentially in using a narrow jig with parallel faces of a material fairly transparent to X-rays. This is placed transversely in the X-ray beam, filled with the sample to be tested, jigged and examined with a fluorescent screen or recorded on a photographic film. As the cell is of uniform thickness the picture will be slightly and uniformly mottled if the contents are clean, pure coal, but will show darker spots where ash is present. The appearance of such a sample of washed pea coal before jigging is illustrated in Fig. 8*a*. Even

[11] C. Norman Kemp: *Op. cit.*
[12] Wm. McLaren: *Op. cit.*

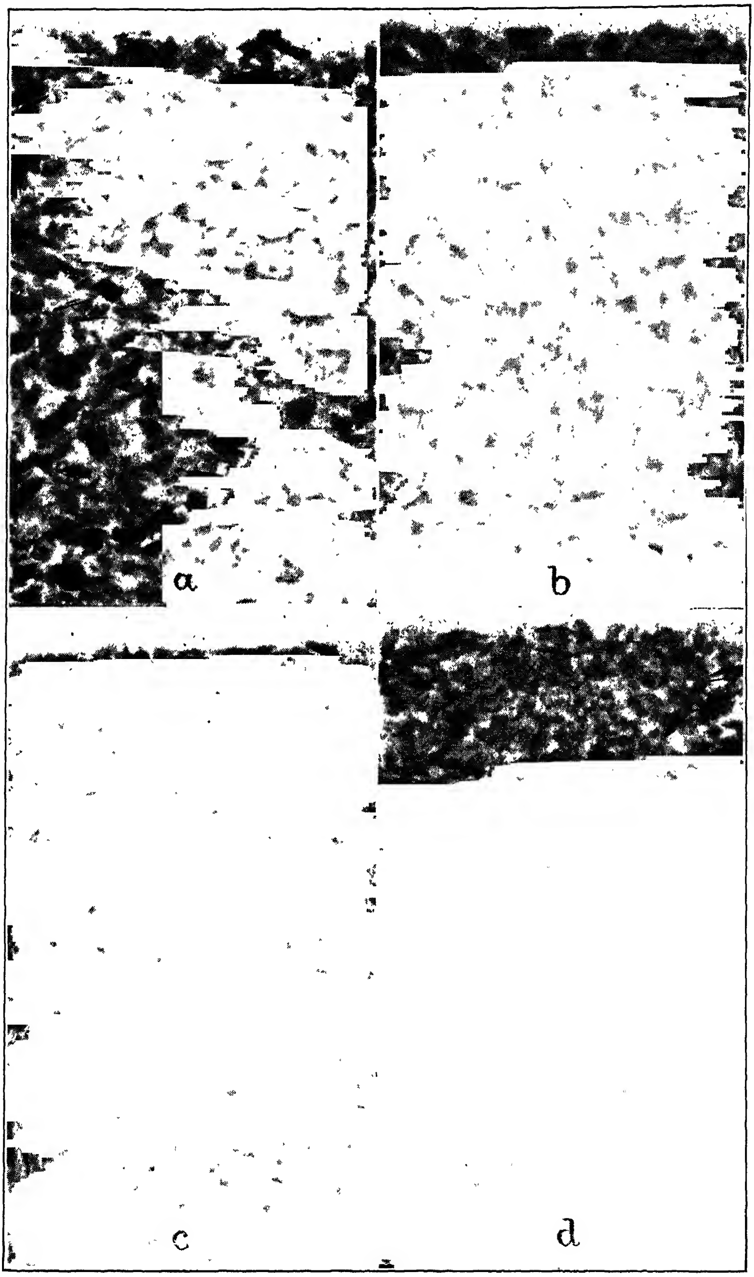

FIG. 8.—RADIOGRAPHS OF WASHED PEA COAL AND ASH. *a.* SAMPLE BEFORE JIGGING. *b.* SAME SAMPLE AFTER JIGGING. MAJOR PORTION OF ASH IS CONCENTRATED AT BOTTOM. *c.* ASH OR DISCARD BEFORE JIGGING. *d.* DISCARD AFTER JIGGING.

the most inexperienced can detect a considerable amount of ash still present. Fig. 8*b* shows this sample after jigging, with the major portion of the ash concentrated at the bottom. The occasional dark flecks in the coal are evidently due to streaks of ash material within the lumps of coal, constituting the so-called intrinsic ash. The rest of the coal was removable but had not been taken out. Fig. 8*c* shows the appearance of a sample of the ash or discard originally removed, and Fig. 8*d* shows this sample after jigging. Evidently an appreciable quantity of good coal was being passed into the discard.

Coördination between Radioscopic Analysis and Ash-characteristic Curves

A detailed description of the method of preparing and interpreting the Henry or ash-characteristic curves is outside the scope of this paper. It is treated at length in the paper by McLaren.[13] Briefly, the sample is jigged in a suitable hand-jig until "effectively arranged in an ideal washing-bed." Layers of this bed are removed successively, dried, weighed and incinerated. The percentage ash in each layer is then plotted against the percentage weight of the layers, reckoned downward.

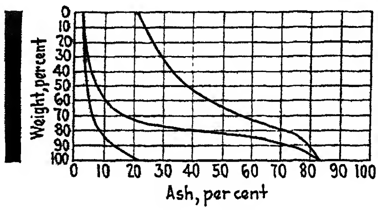

Fig. 9.—Ash-characteristic curves for high-ash coal before washing.

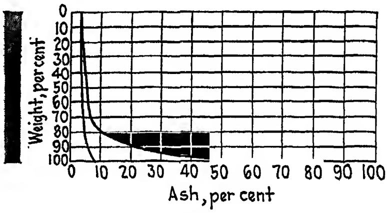

Fig. 10.—Ash-characteristic curves for high-ash coal, washed.

In Fig. 9 the middle curve represents the actual ash content of the corresponding layers. The left-hand curve is derived from this by dividing the total area under the ash curve down to a certain level by the weight of material down to that level. It thus represents the average ash content of the portion of the sample above the level in question. The upper end of this curve coincides with the upper end of the ash curve and represents the intrinsic ash. The right-hand curve is obtained similarly by dividing the area above the ash curve by the weight of material below the level. It consequently represents the average ash content of the lower portion of the sample. The lower end of this curve coincides with the end of the ash curve and represents the removable ash. The upper end of this curve and the lower end of the left-hand curve both represent the average ash content of the whole sample. The radiographic record of this bed is shown on the extreme left for the sake

[13] Wm. McLaren: *Op. cit.*

of comparison. The coördination between the top of the ash zone in the radiograph and the horizontal portion of the curve is notable. Fig. 10 depicts the corresponding curves and radiograph for the coal after washing.

Practical Application of Radioscopic Analysis

To point out the possible uses of these methods in actual practice it is reasonable to quote from a recent letter by Mr. Kemp:

> Among the radiological applications which have been developed by myself and my colleagues during the past few years, the following may be of interest:
>
> 1. Examination of bore cores as auxiliary to usual test methods.
> 2. Fundamental surveys of coal for the determination of washing characteristics.
> 3. Examination of screened coal and fines as to the concentration of dirt in particular size-categories.
> 4. Examination of washery products by laboratory jig test.
>   *a*. The determination in the washed product of the remaining free (removable) dirt in excess of an allowable percentage.
>   *b*. The determination in the discard of the amount of good coal in excess of an allowable percentage.
> 5. Examination of the general character of the coal in specimens of an average thickness of about 2 in., or, where necessary, in specially-cut slices of a uniform thickness of about ½ in.
> 6. Control of coal-washing, based on the results of prior fundamental examination.
> 7. The stereoscopic examination of cokes and products of carbonization as to structure and free-ash distribution.
>
> The feature of these tests is their rapidity and the precision of the information they provide. They furnish also a visual interpretation of analytical data, and have an additional value for purposes of comparison and record.
>
> These tests valuably supplement the information which coal users usually determine for themselves, *i. e.* average ash-content and calorific value, because they indicate what proportion of the ash so determined could and should have been removed at the coal-washery, and the possible savings are sufficiently obvious.

Method of Diffraction Analysis

X-rays can do more than disclose the distribution of materials in this way. In 1912 and 1913 the work of Friedrich, Knipping and Laue[14] and the Braggs[15] showed that X-rays are electro-magnetic vibrations similar to light waves but so much shorter that their wave-lengths are of the same order of magnitude as the diameters of atoms, and that they obey many of the laws of visible light. In particular they showed that these rays give rise to diffraction phenomena when incident upon an orderly array of atoms such as exists in a crystal, just as visible light does when incident upon an orderly array of small obstacles such as the threads of an umbrella-cover. And, just as the pattern observed when a distant light is

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[14] Friedrich, Knipping and Laue: Interference Phenomena with X-rays. Ber. bayer Akad.Wiss. (Math. Phys. Kl.) (1912) 303.

[15] W. H. Bragg and W. L. Bragg: The Reflection of X-rays by Crystals. *Proc.* Roy. Soc. A. (1913) 88, 428.

viewed through the umbrella depends upon the character of the light and the size and arrangement of the threads in the cloth, so the pattern produced when X-rays traverse crystalline material depends upon the character of the X-rays, the size and arrangement of the atoms in the crystals, and the size and arrangement of the crystals themselves.

The method was extended from examination of single crystals to the study of powdered crystals and aggregates of fine crystal grains by Debye and Scherrer[16] and A. W. Hull.[17] In the modified form of the powder method, now the usual practice in this country, the strongest of the characteristic wave-lengths from a molybdenum target is used to

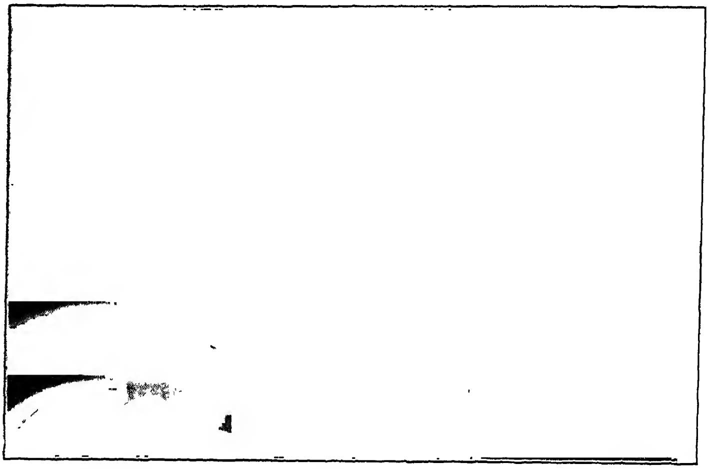

FIG. 11. FIG. 12. FIG. 13.

FIG. 11.—DIFFRACTION PATTERNS OF GRAPHITE AND ANTHRACITE.

FIG. 12.—DIFFRACTION PATTERN FROM FRAGMENT OF ANTHRACITE WITH CLEAVAGES AT 45° TO HORIZONTAL.

FIG. 13.—DIFFRACTION PATTERN OF ASH FROM BITUMINOUS COAL.

illuminate a specimen having a predetermined portion lying on the axis of a cylindrical surface. A photographic film is spread on the surface and the pattern is recorded as a series of lines or bands, one for each set of parallel planes that exists in the atomic arrangements present in the material under examination.

Fig. 11 is a pattern from the author's collection and shows powdered anthracite on the right, compared with a standardized sample of graphite on the left. The broad, diffuse bands indicate that the carbon material is in a state of colloidal dispersion, probably in a hydrocarbon binder of low volatility. The sharp faint lines in the coal pattern

[16] Debye and Scherrer: Interference of X-rays Using Irregularly Oriented Particles. *Phys. Zeits.* (1918) **17**, 277 and (1917) **18**, 291.

[17] A. W. Hull: A New Method of X-ray Crystal Analysis. *Phys. Rev.* (1917) **10**, 661.

are due to the ash, and if compared with the patterns of calcite, pyrites or other possible ash constituents would disclose the character of the ash. These ash lines are somewhat more pronounced in Fig. 12, which is the pattern from a small lump of the coal. Fig. 13 is the pattern of the ash from a sample of bituminous coal.

Close comparison of the negatives shows that it contains other constituents than those discernible in the anthracite. The lump used in

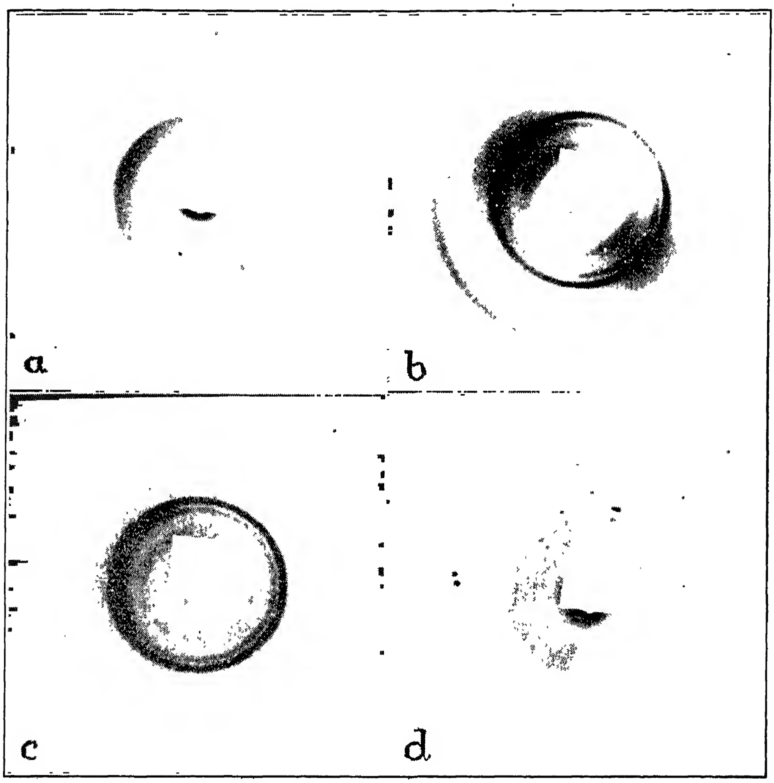

FIG. 14.—"PINHOLE" PATTERNS FROM COAL AND GRAPHITE; PRESSURE AXIS DOWNWARD FROM LEFT TO RIGHT: *a*, ANTHRACITE; *b*, COMPRESSED GRAPHITE; *c*, ARTIFICIAL GRAPHITE FROM COMPRESSED BITUMINOUS COAL; *d*, FRAGMENT OF BITUMINOUS COAL.

making Fig. 12 had its cleavage planes at 45° to the slit limiting the X-ray beam. The greater intensity of the left end of the first band and the right end of the second suggests that the colloidal particles are more or less flaky, like embryo flakes of graphite, and have been preferentially oriented to some extent by pressure in the coal-bed. This is borne out by Fig. 14*a*, in which the pattern due to a fine circular beam has been recorded as a series of concentric circles on a film at right angles to the

beam. The greater intensity of the first band in the 45° direction is marked. The extent to which such preferred orientation can be developed is illustrated in Fig. 14*b*, the pattern from a sample of the standardized graphite which has been submitted to a pressure of several tons per sq. in. It is interesting to note that in Fig. 14*c*, the pattern from bituminous coal similarly compressed, and in Figure 14*d*, the pattern from a fragment of bituminous coal, there is no such evidence of preferred orientation.

Practical Applications of Diffraction Analysis

Possible applications of diffraction analysis to coal problems will undoubtedly suggest themselves to the reader. Among those offering promise may be mentioned:

1. Determination of the state of combination of the elements comprising the ash.
2. Determination of the relative proportions of the various ash constituents.
3. Study of the modifications in the ash composition during combustion in the furnace.
4. Fundamental study of the carbonization characteristics of coals.

DISCUSSION

A. H. Emery, Pittsburgh, Pa.—What is the maximum thickness of pieces that may be photographed?

A. St. John.—Coal is principally carbon, with some oxygen and some hydrogen; normal flesh tissue of the human body is principally carbon, with some oxygen and some hydrogen. Although I have never examined a coal bed to find the thickness through which I could get a picture, I have made pictures through very fat men without any difficulty. I am, therefore, convinced that we can get good pictures through 2 ft. of coal without any difficulty.

A. H. Emery.—What voltage is required for these pictures?

A. St. John.—The milliamperage does not make much difference. I use whatever I think will be safe for the tube under the voltage conditions, which is about one-half what the manufacturers of the tube indicate. I prefer to take twice as long to get a picture and have the tube last about 20 times as long. If the absorption curve is plotted, with respect to voltage for carbon, it begins to flatten out at about 50 to 60 kv., so there is no reason for using 200 kv., which would involve a threefold change in cost of equipment, while the change in penetrating power

would be about 20 per cent. I would use the same conditions under which we work on the human body, and am basing my calculations and design of coal analysis and coal control equipment on that very fact.

A. H. Emery.—Were the pictures shown in Figs. 11, 12 and 13, taken by the Debye and Scherrer method, using powdered coal?

A. St. John.—By a modification of the crystalline-chaos method, which we should call the Hull method. Hull did that work independently of Debye and Scherrer, who published their results first in an obscure place, in Germany, in 1915 and in a place that was accessible to the general public in 1916. Hull disclosed to me the essence of his method in 1915 and I used it in 1916. The specimens that Hull uses are packed in a little glass tube less than 1 mm. inside diameter. I take a weighed amount of the sample of any particular shape or size I desire, even blocks of coal or of compressed carbonaceous material that have been molded with binders under high pressure and then baked in an electric furnace to make artificial graphite. Then I saw out a section of the proper size, set it up in the beam, and have all my samples just alike, weighing the same amount, containing the same amount of carbon, and with any orientation I may want. In my modification of the powder-analysis method, we can go much beyond some of the results that are obtainable by Hull's original method, but the basis of the method is essentially the same.

A. H. Emery.—What was the exposure for Figs. 11, 12 and 13?

A. St. John.—About 15 or 20 hr.; 25 milliamperes with the regular G. E. tube.

Mining Methods in Grass Valley District, California

By J. A. Fulton,* Reno, Nev., and A. B. Foote,† Grass Valley, Calif.

(New York Meeting, February, 1926)

Gold was discovered in the Sierra Nevada by J. W. Marshall on Jan. 2, 1848. The town of Grass Valley soon sprang up and contained several stores in 1849; but the population of the town has always reflected the fluctuations in quartz mining. Gold-bearing quartz was found on Gold Hill in September 1850, and in the next two years most of the quartz-bearing veins in the vicinity were located.

At first, the quartz was crushed in mortars with spring poles; the Huff Co. took out $20,000 in this manner during the winter and spring of 1851. The first mill was put up in January, 1851. In 1866, there were in Grass Valley 248 stamps, crushing 71,420 tons of ore with an average yield of $30 to $35; in Nevada City, there were 142 stamps, crushing 14,200 tons, with an average value of $25. But in 1918, only two properties in this region, the Empire and the North Star, were producing. Since then, there has been a revival of mining in the district and the following properties are being opened up: The Golden Center, Grass Valley Gold Mines Co., Idaho-Maryland Mines Co., while the Sultana is kept unwatered and work is going on in a half dozen small properties.

Mining Claims

At the first meeting of miners in Grass Valley, Jan. 13, 1851, a plot 30 by 40 ft. was allowed for a claim, the boundaries in all cases being perpendicular. In 1852, the quartz miners in Nevada County decided that "each prospector shall hereafter be entitled to 100 ft. on a quartz ledge or vein and the discoverer shall be allowed 100 ft. additional. Each claim shall include all the dips, angles, and variations of the vein." This rule was in force until replaced by the Federal mining laws of 1872; these allowed quartz claims to be 1500 ft. along the vein, with all the dips, spurs, and angles, and to have a width of 600 ft.

The holdings of the principal companies consist of these early patented mining claims; standard claims, as authorized by the Act of 1872, both patented and unpatented; agricultural tracts, both patented and

* Director, Mackay School of Mines.

† General manager, North Star Mines Co.

unpatented, carrying in some cases both surface and mineral rights and in other cases mineral rights only; and mineral rights under town lots, streets, etc. As a result, there is no uniformity in the size or shape of the principal mineral tracts. All titles are held in fee.

Geology

The Nevada City and Grass Valley district is in the west central part of Nevada County, Calif., on the western slope of the Sierra Nevadas. It is in the higher foot hills, near where the igneous rocks give place to the slaty sedimentary rocks of the middle range. East of this sedimentary belt, the granite rocks of the highest parts of the range begin.

The region of complicated structure begins at Rough and Ready, 5 miles west of Grass Valley, with amphibolites and gabbros; then comes a belt, ¼ to 1½ miles wide, of sedimentary clay slates and siliceous slates. This enters the Grass Valley tract near the North Star mine. A branch of these slates traverses the northeastern part of the Grass Valley tract. Between Grass Valley and the main clay-slate belt is a body of diabase and porphyrite about ¾ mile wide. East of this diabase is a narrow body of granodiorite 5 miles long, the northern end of which is in Grass Valley. East of this is the porphyrite and diabase area of Osborne Hill.

In the Empire and North Star mines, the veins in the upper levels are in the diabase; at varying depths, however, they pass into the granodiorite. After entering the granodiorite, the value, width, and character of the veins change but little. For many years, production has been from the veins in this rock.

In the other mines, most of the veins are in the diabase. In the Eureka-Idaho-Maryland, a diabase dyke 20 to 30 ft. wide intrudes the serpentine and the Eureka-Idaho-Maryland vein is on the foot wall contact of the diabase with the serpentine. This contact has been consistently to the lowest levels yet reached—2300 ft. vertically. Finally, after the intrusions of the granodiorite, dynamic forces, acting on the mass with different intensity and in different directions, produced fissure systems in which auriferous solutions ascended and deposited their contents. This forming of the mineral veins was the last phase of the Mesozoic revolution of the Sierra.

General Description

The elevation of the district varies from 2000 to 3000 ft. above sea level; the area consists of gently rolling hills, heavily wooded with white pine, spruce, cedar and oak. The timber is all new growth and is well suited for mining purposes. All underground timber is round, except that used in shafts, etc. Pine is used in the stopes and spruce for develop-

ment. The usual sizes are from 8 to 12 in. in diameter; timber of this size will grow in less than 40 years so that the supply is practically inexhaustible. The climate is ideal to operate in. The extremes of temperatures are from 10° to 90° F.; the normal rainfall is about 60 in. There is a little snow each winter but not sufficient to be troublesome.

The labor supply is exceptionally good, over 80 per cent. is of the English-speaking races, and many of the men own their homes. Although the supply is not always adequate, conditions are better than in other districts at all times. A local organization, known as the Mine Workers Protective League, has been in existence for over 4 years and the relations between the league and the mining companies are exceptionally satisfactory.

The district is connected by the Nevada County Narrow Guage Railroad to the Southern Pacific at Colfax, 16 miles; a paved highway from Grass Valley to Auburn, 25 miles distant, connects with the Lincoln Highway.

The conditions underground are about as favorable as the surface conditions. The ground stands well, requiring practically no timbering, except in the Idaho-Maryland, where the serpentine foot wall requires extensive timbering. The mines are cool and the water handled is very small, considering the miles of underground workings. The future does not present any pumping difficulties so far as is known, in fact, the mines are much dryer in the lower levels than in the upper. The district is served with electric power, the cost of which is 9 mills per kilowat-hour to the larger mines. The service is excellent in every respect. In addition to the electric power supply, some of the older mines use water power to a limited extent to operate air compressors and mine pumps.

[Acknowledgment

Much of the information pertaining to geology and history was taken from the report of Waldemar Lindgren, "The Gold-quartz Veins of Nevada City and Grass Valley Districts," published by the U. S. Geological Survey.

North Star Mine[1]

This mine has had various owners since its opening in 1850. In 1886, the main incline shaft was down 1500 ft. and a 40-stamp mill, with a capacity of 75 tons a day, was in operation. At that time, it was owned by the North Star Mining Co., whose holdings consisted of the North Star patented claim, which was made up of a number of small claims, and included 3140 ft. of the outcrop of the vein. In 1899, the property

[1] Written in 1922.

was transferred to the North Star Mines Co., and adjoining tracts were purchased until the company owns 1430 acres, both surface and underground, and the mineral rights of about 400 acres. The acquisition of

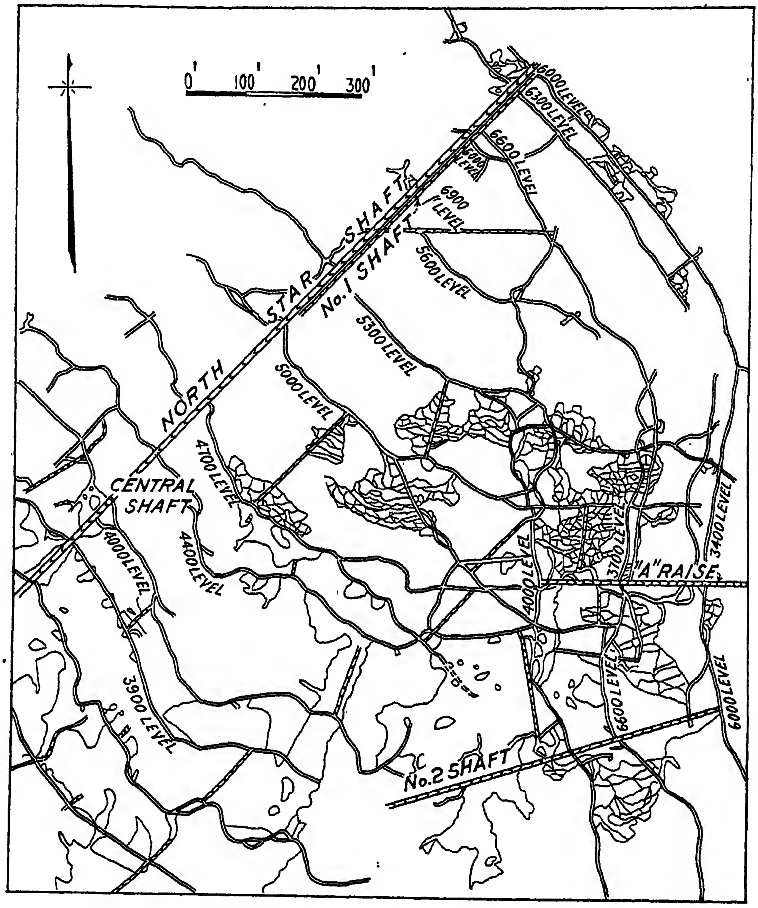

FIG. 1.—WORKING MAP OF PART OF NORTH STAR MINE.

the adjoining land was necessary, as the mine is being worked beyond the extralateral rights pertaining to the original North Star patented claims.

Sampling and Estimating

In general, the ore is hard quartz, which sometimes is frozen to the walls. It varies in thickness from 1 in. to 5 or 6 ft., but averages 16 in. If very rich, a thickness of 4 in. may be profitably mined. The vein is flat, averaging 26° from the horizontal. Sampling is done by taking a cut at right angles to the walls.

No regular method of sampling mill heads is in use, because of the erratic results obtained. The tailings from the stamp mill (cyanide heads) and the cyanide plant (final tails) are thoroughly sampled and assayed daily. A close estimate of the daily production is made from the ounces of quicksilver fed in the mill each 24 hr. In the cyanide plant, the number of tons of solution precipitated each day is measured by a meter, and the value of the solution is determined by assay.

Recently, some diamond drilling has been done, but nearly all exploring has been by drifts, shafts, crosscuts, etc.

Maps are brought up to date each month, and models each year. Areas stoped and developed are measured on the maps with a planimeter; this gives the usual unit figures for production and percentage of profitable ground.

Estimates are based entirely on experience. Development is carefully inspected and the various areas classified as ore or waste. The production and tonnage expected from the areas classed as ore are based on units per square foot, which in turn are based on the records of the past twenty years. Generally, the estimates are low as the areas classed as ore prove to be larger than estimated. No percentage of variation can be given.

History of Mining Methods

The orebodies are very irregular in shape and occurrence; about one-third of the area of the vein explored has proved profitable. No reason, such as intersecting veins, dykes, or faults, has been discovered for one portion of the vein being ore and another barren.

The minimum angle of flow of broken ore is 45° on the foot wall or 40° in a chute with a plank bottom. The angle can be reduced to 35° if a small stream of water flows over the bottom of the chute.

As the average thickness of the vein is only 16 in., while the average width of stope is 42 in., it is necessary to handle about twice as much waste as ore. The rock is hard; it requires 1.6 lb. of powder and 3 ft. of hole drilled for each ton of ore (and the resultant waste).

At first, levels were run from 60 to 100 ft. apart on the vein. The ore was shoveled, through "passes," into cars of 1600-lb. capacity, which were hoisted up the main incline shaft and run direct to the mill. At present, drifts are run 300 ft. apart on the vein and cars of 1-ton capacity are used. These are generally hauled by mules, from seven to nine cars forming a train. These cars are loaded from chutes built on the levels, and dump into bins in the main shaft, from which the skips are loaded.

The chutes are filled by cars of 1500-lb. capacity, into which the broken ore from the stopes is loaded by shovels or scrapers. These cars run on temporary tracks built along the face; as the face recedes, these tracks are moved close up. The loaded cars are lowered into the chutes by

means of a wire rope, which passes around three sheaves upon which a brake acts. The cars are dumped automatically, and as the loaded car descends it raises an empty one.

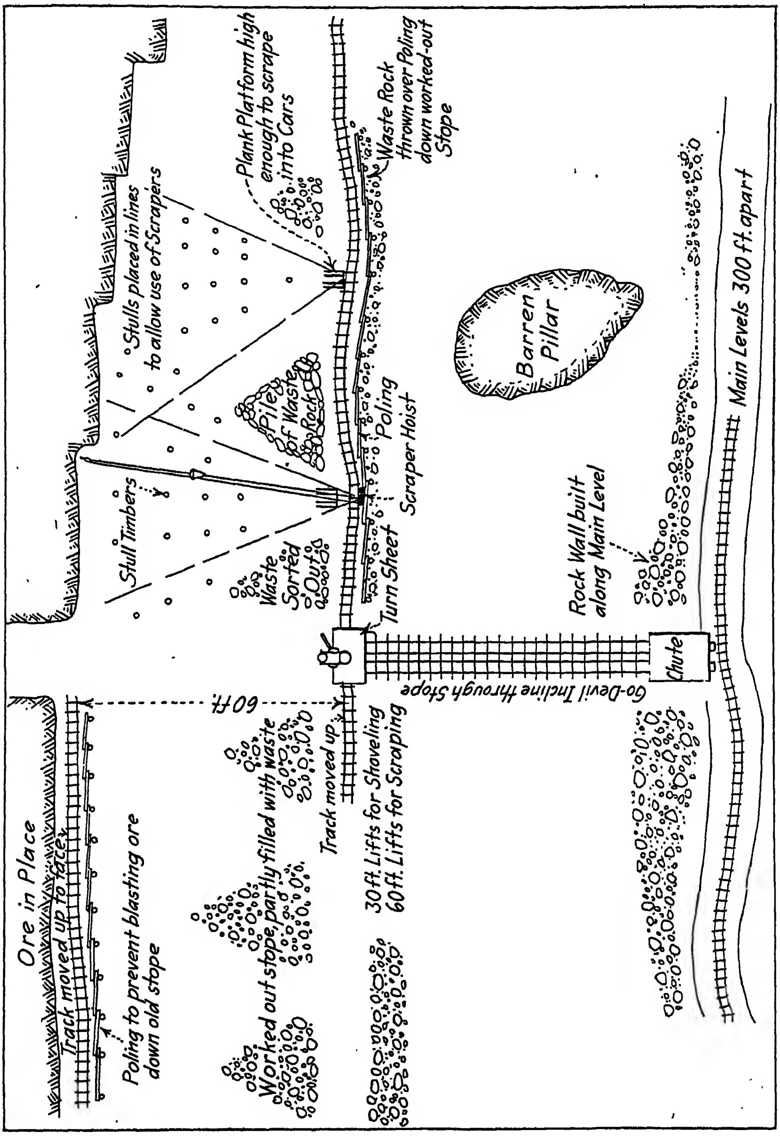

FIG. 2.—HANDLING ORE ON STOPES WITH GO-DEVIL PLANES AND SCRAPERS

As the ore is loaded into these cars, the coarse waste is sorted out and thrown down the stope; about one-half of the waste broken fills the stope, the rest of the waste is then hoisted to the surface for disposal. At the

surface, the ore is again sorted and about 25 per cent. of the material hoisted is discarded as waste; nevertheless about 60 per cent. of the material that reaches the mill is waste.

Later Development

Since 1922, another vein of the North Star mine, dipping in the opposite direction to the North Star vein and crossing it at the 6300 level, has been developed by an incline shaft underneath the North Star shaft to a distance of 1000 ft. down the dip of the vein below the 6300 level. It is the intention to sink the vertical shaft and to continue sinking this incline shaft until the two meet. This meeting point will be about 2300 ft. down the incline from the 6300 level (making it the 8600 level) and about 2000 feet vertically below the 4000 level, where the vertical shaft cuts the North Star vein. The vertical shaft will then be 3600 ft. deep. It is now expected that the two shafts will meet toward the latter part of September, 1926.

Mine Openings

The main incline shaft is 6300 ft. long and has an average pitch of 26°; its vertical depth at the bottom is 2400 ft. The vertical shaft is 1600 ft. deep and meets this incline shaft 4000 ft. from the collar. Both shafts have three compartments; the two skip compartments are 4 ft. 8 in. by 5 ft., the third compartment, which is 5 by 5 ft., is used for the tools and material cage or truck, pipes, etc. The inclined shaft is timbered only in a few places; the vertical shaft is timbered with 10 by 10-in. wall and end plates with 8 by 10-in. dividers. Timber is necessary to support guides, rails, and pipes. The ground stands without timbering. In the vertical shaft, for 40 ft. below the collar, the timbering is gunited on wire lathe; below that point the timbering is always wet and lasts indefinitely. It has now been in place 22 years. All hoisting is done through the vertical shaft, the skips making a vertical turn and traveling down the incline shaft to reach the bottom of the mine.

Underground Development

Main levels, 5 by 7 ft. in cross-section, are run at intervals of 300 ft. on the vein. Water is conveyed in open ditches running along one side of the track. As the drifts are run, 2½-in. pipes for compressed air are laid; sometimes additional 4-in. pipes are laid for particularly large stopes. Ventilation pipes are usually 8-in. though sometimes 6-in. pipes are used.

Drifts are not timbered as a rule. In stopes, 8-in. round posts are put in to hold up loose rocks where required. Some timber is dipped in

"Cabots Conservo" before being sent underground, but only a very small proportion (used in main stations etc.). No timber is framed. Some poling in the stopes, used to prevent ore from being blasted down the stopes below the intermediate tracks, is used more than once.

One 2½-ton storage-battery locomotive, with a drawbar pull of 800 lb., is used on a 3000-ft. haul. This locomotive pulls ten cars with a capacity of 3600 lb. of ore each. The particular level on which this locomotive is used was driven long ago and in places has a grade of over 2½ per cent. in favor of the load. On account of this excessive grade, the capacity of the locomotive is limited by the number of empty cars it can pull back. Practically all other tramming is done by mules, hauling 18-cu. ft. side-dump cars equipped with Hyatt roller bearings. The standard track is 16-lb. rail, 20-in. gage, laid on a grade of 0.8 per cent.

Chutes on levels have an average capacity of 20 tons; bins in stations hold 75 tons. Skips are loaded through chutes with arc gates; rack-and-pinion gates are also installed for use in case of a "run." On account of the flat incline, the upper end of the skip to be loaded is lifted 3 or 4 ft. off the rails, so that ore from the chute will run to the bottom. This lifting is done by an air lift.

The main hoist is run by compressed air; the cylindrical drums are 9 ft. in diameter by 6 ft. wide. It is capable of handling 4 tons of ore at 1200 ft. per min. The ore skips travel in both the vertical and the incline shafts. Drums have 5000 ft. of 1⅛-in. rope wound in three layers. When shifts are changed, double-decked cages are substituted for the skips. These cages travel in the vertical shaft only; at the bottom the men ride on a truck and are lowered down the incline shaft by an electric hoist at the station at the bottom of the vertical shaft.

Waugh No. 50 or Sullivan D. P. 33 (mounted water-type) drills are used for stoping; Ingersoll-Rand No. 148 or Denver Rock Drill Co. Turbo drills are used for drifting. In all types of drills, 1-in. quarter-octagon steel without lugs or collars are used. A four-point bit with enlarged center hole is used. Miners drill for two shifts and blast at the end of the afternoon shift (12:30 a.m.); 35 per cent. and 40 per cent. 1¼ in. diameter gelatine is used. There is no particular arrangement of holes.

Pumping

The water originating in the upper 2400 ft. of the mine is, as a rule, handled by a Cornish pump, run by water power. This pump was built about 1884 and has been running ever since. One man looks after the pump during the day shift, during the other 16 hr. it runs without attendance. An electric plunger pump with a capacity of 500 gal. per min, is at the 1400-ft. station, and an electric turbine pump with a capacity of 300 gal. per min. is at the 2400-ft. station. These are for emergency and

when there is an excessive inflow of water. At the 4000-ft. station, pumping direct to the drain tunnel, there are two ten-stage turbine pumps with a capacity of 400 gal. per min. each, against a 1400-ft. head. A five-throw plunger pump with a capacity of 550 gal. per min. is now being installed alongside these pumps. Smaller turbine pumps on the 5300-ft. and 6300-ft. levels pump to the 4000-ft. level. In addition, there are air pumps to handle the water in case of a breakdown of the electric power supply.

Air Compression

Compressed air is furnished primarily by a 2400-cu. ft. and a 1200-cu. ft. compressor direct-connected to Pelton water wheels. During certain periods, it is necessary to run a 1200-cu. ft. compressor direct-connected to a synchronous motor and a 900-cu. ft. compressor belted to a motor. A 1200-cu. ft. compressor, belted to a motor, is used in case of a failure in the water supply, and a 300-hp. electric motor can be belted to one of the water-driven compressors in the event of such a failure. Water power costs about $36 a year per horsepower, electric power costs about $68 a year per horsepower.

Records of Unit Production, Exclusive of Mill and Cyanide Total Production

In a 31-day month, 10,000 short tons are crushed. This is equivalent to 13,500 tons mined and sent to the ore-sorting plant, as 3500 tons are sorted out as waste.

Mining is done by day labor. Development work is done by contract, based on feet of advance.

In stopes, the production is 0.95 tons per hour per miner and 0.9 tons per hour per shoveler, which includes shovelers stowing waste; or 0.47 tons per hour per man in stope.

On development of ore, the tons per man per hour is negligible. On development in rock, the production is 1 ton per man per hour; and for all underground labor, including men on development work, the production is 0.215 ton per man per hour.

The production for all surface labor, exclusive of office force, is 1.50 tons per man per hour; and for the whole organization 0.163 tons per man per hour.

Classification of Labor

	Shifts	Per Cent.
Bosses	9	2.68
Pumping	7	2.08
Sorting	24	7.15
Timbering	30	8.93
Breaking rock	70	20.83
Shoveling	70	20.83
Tramming	22	6.55
Underground hoists	14	4.17
Tool boys	10	2.97
Sundry	15	4.47
Development	28	8.34
Total underground	299	89.00
Blacksmiths, mechanics, electricians	10	2.97
Watchmen and miscellaneous	7	2.08
Hoisting engineers	4	1.19
Compressors	3	.89
Tool sharpeners	9	2.68
Carpenters, and sawyers	4	1.19
Total surface	37	11.00

Labor Turnover

With 380 men employed at one time, about 425 names are on the pay roll during the month. During the first six months of 1922, the total labor cost was 57.5 per cent. of the total; while the mining labor was 64.2 per cent. of the mining cost.

Records of Units of Supplies

For operating, 1.63 lb. powder per ton is used; 1.75 lb. when powder used in development is included.

Timber used is 0.5 lin. ft. of poles (average 10 in. diameter) per ton; this includes poles sawed into lumber and lagging. No lumber is bought.

The total of 3.523 hp. per ton daily capacity, including that used in mill and cyanide, is used as follows:

Mining	0.962
Haulage and hoisting	0.586
Pumping	0.670
Ventilation	0.385
Lighting and miscellaneous	0.100
Mill and cyanide and sorting	0.800
	3.523

The compressed-air requirements are as follows:

Hoisting	3,300 cu. ft. free air per ton
Drills	5,600
Ventilation	2,200
Total	11,100 cu. ft. free air per ton

The supplies used in mining, exclusive of powder, timber and power, form 16 per cent. of the total cost, exclusive of milling; also 16 per cent. of the total cost of production was for all supplies, except powder, timber, and power. For the first 6 months of 1922, supplies formed 24.4 per cent. of the total cost and mining supplies formed 25 per cent. of the total mining cost.

Ore Recovery

Ore recovery is probably, over 90 per cent.; the only loss is spilling from cars, etc. The foot wall of stopes is swept with brooms before the stope is abandoned.

Safety and Welfare Work

Every reasonable precaution is taken in the interest of safety. Enough men are trained in first aid to have at least one man so trained in each stope on each shift.

But little welfare work is done. All employees live in town. The companies, by advancing purchase price and allowing payments to extend about 10 years, are encouraging some of their employees to own their own homes.

DISCUSSION

W. F. Boericke, New York, N. Y.—There has not been much change in the methods of mining during the last 20 years, for these methods have been found the most applicable to a mine in this country. One change is the use of scrapers and I wish more detailed information on the cost of scrapers had been given.

I have often wondered why the North Star managers did not use gasoline haulage on their long levels; the mine never had the slightest ventilation trouble. In another field, substituting gasoline locomotives for mule haulage on long hauls where there were no heavy grades made considerable saving.

R. M. Raymond, New York, N. Y.—What scraper was used?

G. A. Packard, Boston, Mass.—When I visited the mine 4 years ago, they were using a very light scraper of the hoe type with teeth on one side and flat on the other. My recollection is that it was less than 3 ft.

long and not more than 8 in. high. Much of the vein, as broken, is waste so it is important to leave as much of that in the stope as possible. With a larger and heavier scraper, the miners would not sort out the waste so carefully as when the ore was handled with shovels, as some of the waste would be buried under the large pile. The scraper now used is of the LeClaire type, that is, a piece of angle iron bent at right angles so that the two legs make an angle of 45° with the surface on which it rests.

A. B. Foote (author's reply to discussion).—The Empire and the North Star are about equal in importance and are by far the two largest mines in the district. Each employs about the same number of men; at times the production of the North Star has been larger, but at present that of the Empire is larger. Both handle about the same tonnage.

The State mining laws forbid the use of gasoline locomotives underground, besides there is danger of gasoline fumes accumulating in pockets where there is no ventilation. A storage-battery locomotive is used on a long haul where a large tonnage is handled. Due to the smallness of the vein, the tonnage per unit area is comparatively small, and mule haulage is cheaper than any other kind when the whole output from one level can be taken care of by one driver per shift.

Small scrapers are used not only because of the importance of sorting out waste, but because only a small tonnage is produced per unit area. A large scraper would clean out the whole area within its reach in a short time, necessitating frequent moves and it costs too much to move the large heavy hoist necessary to handle a large scraper from one place to another.

Sharpening and Handling Drill Steels at Franklin

By C. M. Haight,* Franklin, N. J.

(New York Meeting, February, 1926)

The mine blacksmith and drill-steel sharpening shop at the Franklin mine of the New Jersey Zinc Co. is on the surface, adjoining the main shaft. It is a brick building, 51 by 30 ft. inside dimensions, with a corrugated iron roof and concrete floor. About two-thirds of the steels used are brought up through the main shaft; the remainder through two auxiliary shafts and are carried to the shop by team or truck.

At each shaft sufficiently large stocks of sharp drills of various types are kept to replace all the dull steels that may be sent up; while at the shop enough sharp drills are kept to replace the dull steels in the stocks kept at the auxiliary shafts. Though this entails keeping a larger number of drill steels in service than might be required by other methods, there is less hurry in the sharpening shop and the steels have a longer period of rest. If a rest from use enables steel to recover from fatigue strains, this method secures that rest.

The forms shown in Fig. 1 are supplied to each shift boss in books of 100. These are used to requisition sharp drills from the shaft stocks and enable the men at the tops of the shafts to sort out the required sizes and to have them ready to be sent to the various shaft stations. At one of the auxiliary shafts, the dull drills are placed on a platform from which they are dumped into the body of the wagon that takes them to the shop.

Order Slip for Drill Steel

Date	Water Leyner		Stoper		Block Hole			
....	Overall Lengths	No. Pieces	Overall Lengths	No. Pieces	Overall Lengths	Pieces Big	Pieces Small	
Starter...........	4′-6″ to 4′-0″		2′-6″ to 2′-3″		2′-6″ to 1′-6″			
Second...........	7′-0″ to 5′-6″		4′-0″ to 3′-3″		4′-6″ to 2′-7″			
Third.............	8′-6″ to 7′-1″		5′-6″ to 4′-9″		6′-6″ to 4′-7″			
Fourth...........			7′-0″ to 6′-3″		8′-6″ to 6′-7″			
Fifth.............			8′-6″ to 7′-9″		10′-6″ to 8′-7″			
Sixth.............					12′-6″ to 10′-7″			Boss..................

Fig. 1.—Order forms supplied to shift bosses.

* Mining engineer, New Jersey Zinc Co.

SHOP PROCEDURE

At the shop the dull drills are sorted into racks behind the sharpening machines, according to type and length. A count is made at this time and all bent, broken, or plugged steels are laid aside for repairs. Broken drills, if too short for the limits shown on the requisition form, are cut back to the next shorter length or, if too short even for starters, are discarded.

From the racks the drills to be sharpened are placed on a movable table, Fig. 2, which is then wheeled into place at the furnace as shown in Fig. 3. From this position the drills are placed in the furnace and, when properly heated, are passed by the helper to the man at the sharpener.

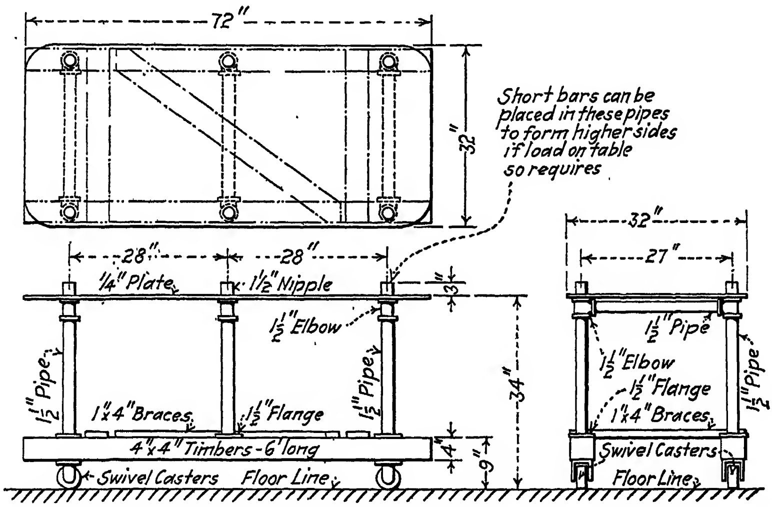

FIG. 2.—MOVABLE TABLE FOR DRILL STEELS.

After the drills are sharpened, they are placed on a second movable table which is moved to position at the Gilman heat-treating machine. After heat treating, the drills are inspected and placed in racks.

No steels are passed through the heat-treating machine until they have cooled from the sharpening heat, which allows forging strains to ease themselves. This is an important precaution.

The two sharpening machines are Sullivan Class A sharpeners, on each of which a Model 10 Waugh drill-steel puncher is mounted. The drills to be punched are held in guides mounted in the place on the clamp provided for the shears, and are thus centered and held tight. Each machine is equipped with a gaging device, adjustable to 1/16 in., with which the double-taper bit with a reaming edge is obtained; but to get this result it is necessary to use a dolly 1/8 in. larger in gage than is required

in the finished bit. Shank gages are attached to the sharpener; different types of drill bushings for testing the diameter of the shanks and a set of ring gages checking the bit diameters, are provided at each machine. Fig. 4 shows a drill and puncher.

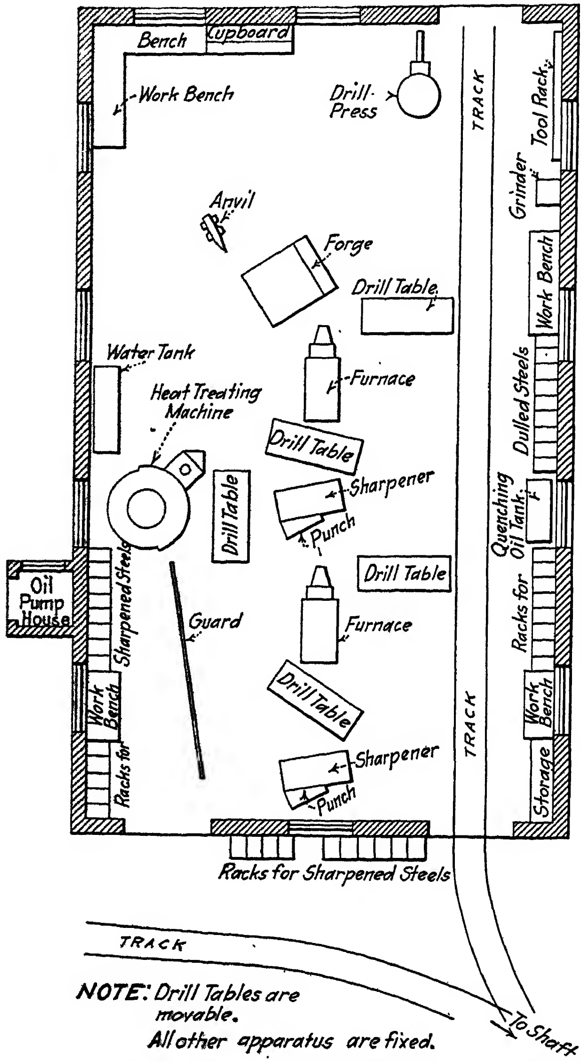

FIG. 3.—SHOP, 51 BY 30 FT. INSIDE; BRICK CONSTRUCTION.

When shanks are made, it is necessary to cool the ends of the steels before upsetting them for lugs and collars. For this purpose a shallow ditch with running water has been placed in the floor, behind the sharpener operators' positions.

The bits made are of the regulation double-taper type. On the 1¼-in. round hollow steels, flat-faced crossbits and the regular Leyner lugged shanks are used. For stoping drills, a ⅞-in. quarter-octagon section is used with raised center crossbits for solid steel, and flat-faced crossbits for hollow steel; straight shanks are used on all stoping drills. Block-hole drills are also made from the ⅞-in. quarter octagon shape but have six-point bits and the jackhammer type of shanks.

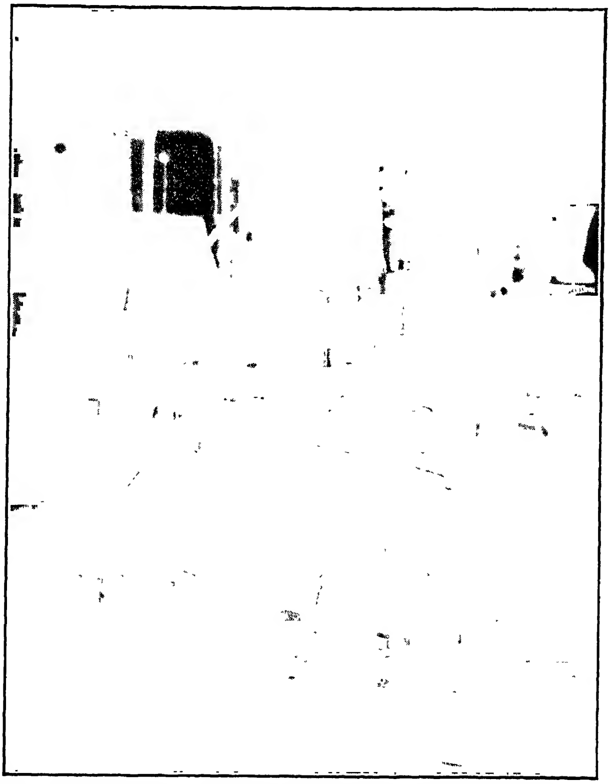

FIG. 4.—SULLIVAN DRILL SHARPENER WITH DENVER DRILL PUNCHER.

Sullivan drill-steel furnaces, burning fuel oil, are used for heating the steels for sharpening. One side only is used; the other is bricked up. The furnaces are run with a reducing flame. These furnaces have been equipped with automatic pyrometer controls which will cut off portions of the oil and air supplies if the furnace temperatures get higher than a set point; when the temperature has been lowered the valves open again. This control is used mainly to prevent overheating, so that if the furnace adjustment is in order, there should be no interruption from a steady heat. Should an interruption occur, it is a sign to the blacksmith to adjust his furnace-regulating valve. Valves

on both the air and the oil lines are necessary with this control, for if only the oil is cut off the flame is changed to an oxidizing one, which is undesirable.

The air for the furnaces is supplied from the mine compressed-air line but is greatly reduced in pressure by proper valves. A hood with a stack extending through the roof is placed over each furnace. In addition, an overhead ventilating system supplies outside fresh air on or near each working position at the furnaces, sharpeners and heat-treating machine.

The fuel oil is stored in an underground tank outside the shop and is pumped up by a reciprocating air pump. The air supply for this pump is from the shop, therefore a failure in the air causes the pump to

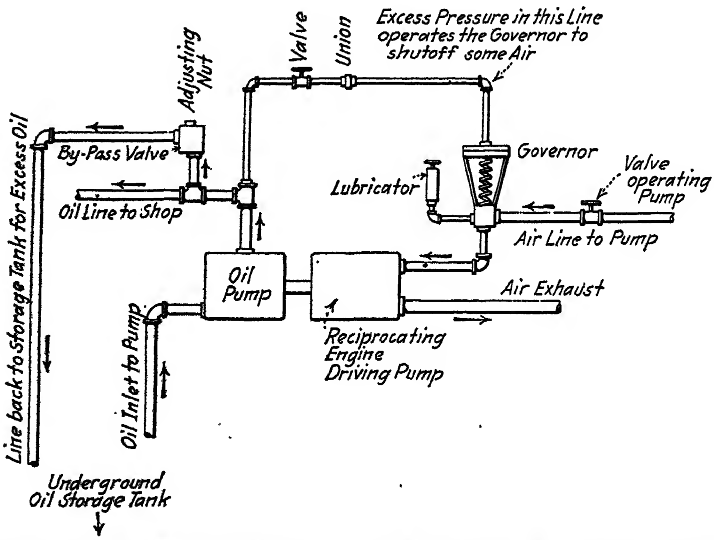

FIG. 5.—DIAGRAMMATIC SKETCH OF OIL-PUMPING SYSTEM. (NOT TO SCALE.)

stop. If a pressure of 15 lb. on the oil line is built up, a bypass valve operates to allow the excess oil to be pumped back into the storage tank; hence any starting or stopping of furnaces does not necessitate a regulation of the pump. A diagrammatic sketch of this installation is shown in Fig. 5. All pipes are under the floor in ditches, which are covered with 2-in. plank laid so as to lie flush with the floor.

HEAT-TREATING MACHINE

A Gilman C. E. 21 heat-treating machine is used to harden the drill bits. This machine has been in use since April, 1922, and has proved very satisfactory. This equipment has only recently been placed on the market; that at Franklin was the first one made and has to some

extent served as an experimental plant. For this reason a more extended description of this device is given than of the better known appliances.

Briefly, the machine comprises an oil-burning furnace and a quenching tank, above which is arranged a circular, stationary carrier track of varied gradient. Carriers for holding the drills are driven along this track by a power mechanism, so that the drills are carried down into and through the furnace and into the quenching tank before they are removed from the carriers and ejected. A profile of this track is shown in the upper part of Fig. 6.

To harden drill steels properly mechanically, certain requirements must be met to obtain uniform results. These are:

1. A furnace in which a constant temperature may be maintained uniformly throughout its length.

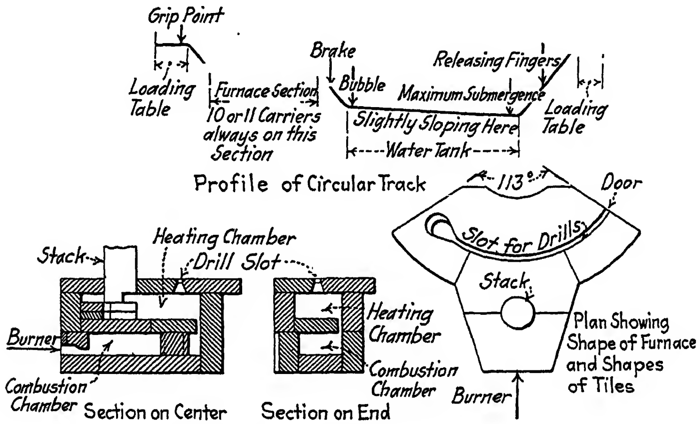

FIG. 6.—SKETCHES OF FURNACE.

2. Automatic means to keep the heat constant at any desired temperature.

3. A definite time interval for the steels to be in the furnace.

4. A uniform path to be followed by the steels, both through the furnace and through the water.

5. All parts of the drill bit to be equally exposed to the furnace heat.

6. Heating to be done in a reducing atmosphere and preferably in a chamber separate from the one in which combustion takes place.

7. Steels should be inserted into the furnace deeply enough to be heated as far back from the cutting edges as is required to counteract the effect of the forging temperatures on the grain size.

8. Transfer from the heat to the quenching water should be done rapidly enough to insure quenching on a rising heat.

9. The formation of soft spots on the cutting edges from the presence of steam bubbles should be prevented.

10. Provision must be made to heat the large gage bits for a longer period than those of small gage.

11. After the cutting and reaming edges have been hardened, further quenching should be so conducted that a gradually blended and toughened structure is obtained back of the cutting edges.

The Gilman heat-treating machine meets these requirements in all respects. In addition to the furnace, quenching tank and circular track

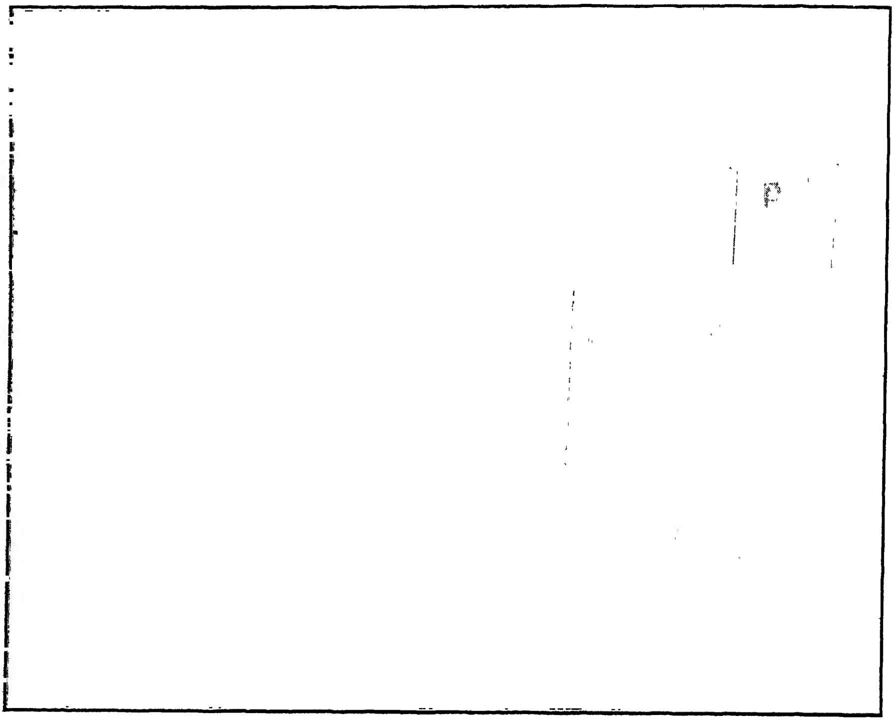

FIG. 7.—HEAT-TREATING MACHINE AT FRANKLIN; EQUIPPED WITH SIGNALLING PYROMETER CONTROL (NOT AUTOMATIC). NOTE STEELS IN FURNACE AND STEEL IN QUENCHING TANK.

mentioned above, the other major parts of the machine are the electric motor, the mechanism for moving the drill-steel carriers along the track, the loading table, and the blower which supplies air to the furnace.

In the center of the track is the operating mechanism, which consists of a central revolving pedestal from which four radial arms extend horizontally. At the ends of these arms are grips which, through means of springs and cams, either move the steel carriers along the track or pass them by, as required. The carriers are equipped with several spring

clips or grips; the lowest one is powerful enough to prevent the steels from slipping down, while the rest serve to hold them in a vertical position.

The reheating furnace is of the semi-muffle type in which the heating chamber is separated from the combustion chamber. Due to the automatic pyrometer control now used, the heating chamber keeps at the desired temperature, generally 150° F. above the upper critical range of the drill steel. A reducing or nonoxidizing flame should be maintained

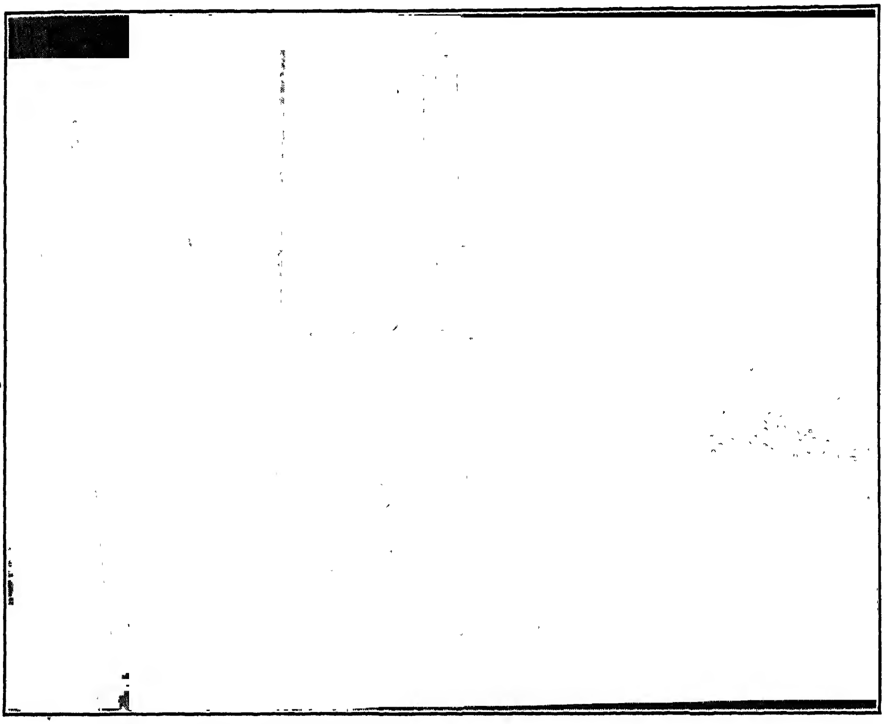

FIG. 8.—HEAT-TREATING MACHINE AT FRANKLIN.

in the furnace. The furnace is placed outside the circular track and is built on an arc conforming to the path through the drill steels travel. The opening through which the ends of the drills are introduced into the furnace is a slot in the top; at the exit end is a steel door, opened by pressure of a steel against it but otherwise kept closed by a weight. The furnace cross-section and plan are shown in the lower part of Fig. 6.

The quenching tank also is built to conform to the arc through which the drill steels travel. At the end near the furnace is placed the inlet pipe, which is flexible enough to allow some adjustment. As the water leaves this inlet pipe, it forms a bubble above the surface of the water

in the tank, and the pipe should be so adjusted that the hot drill bit stops immediately over it, with the water playing against the cutting edges. At the other end of the tank is an overflow pipe, adjustable to height, which regulates the depth of the water in the tank.

Figs. 7 and 8 show the machine at Franklin. Figs. 9 to 12 show the machine as now built.

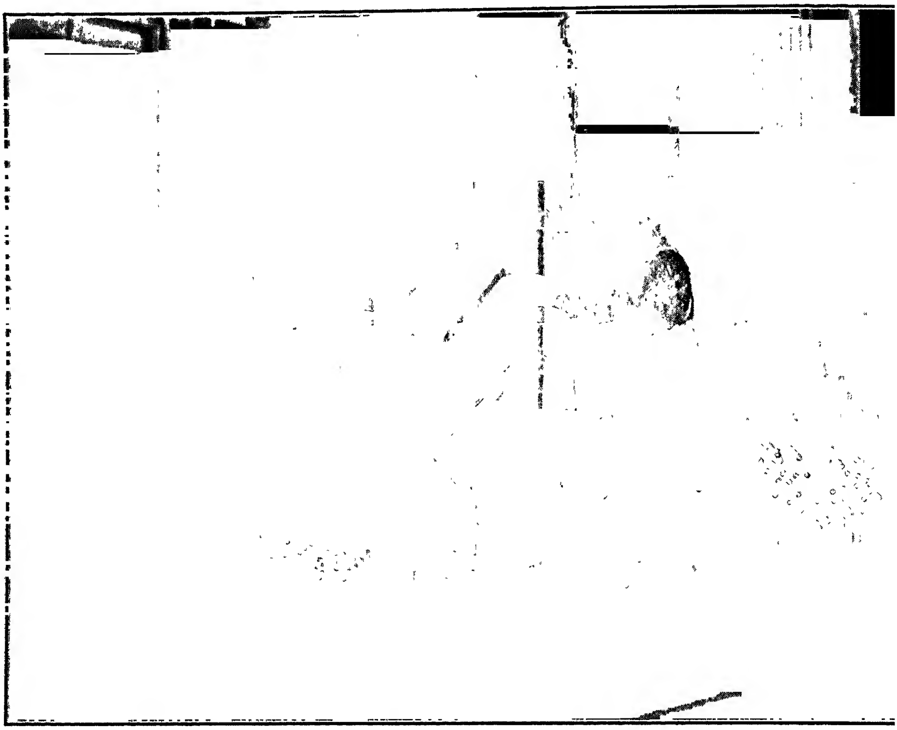

FIG. 9.—STEEL ABOUT TO BE EJECTED FROM CARRIER; OPERATOR READY WITH A STEEL RESTING ON LOADING TABLE, STEELS IN FURNACE, ONE OVER BUBBLE, AND ONE IN TANK.

CYCLE OF OPERATIONS

In the description of the cycle of operations which follows, it is assumed that the reheating furnace is at its proper heat and all adjustments necessary to run the machine have been made, including the proper speed for the bit gage.

A steel carrier (there are 14 of them), gripped by one of the four arms on the central driving pedestal, is brought along the track to the loading table. While passing over the table, the bottom grip is automatically kept open. The operator takes a sharpened but unhardened drill steel

from the pile on the movable table and, while holding it vertically with the bit end down, places the bit on the loading table and then shoves the steel into the carrier, forcing the upper parts of the steel into the clips on that part of the carrier. Thus the steel is assured its proper position in the carrier and in its passage through the machine. As the carrier passes around the loading table, the bottom grip is automatically closed so that it can take hold of the drill steel. Still in the grip of the carrier

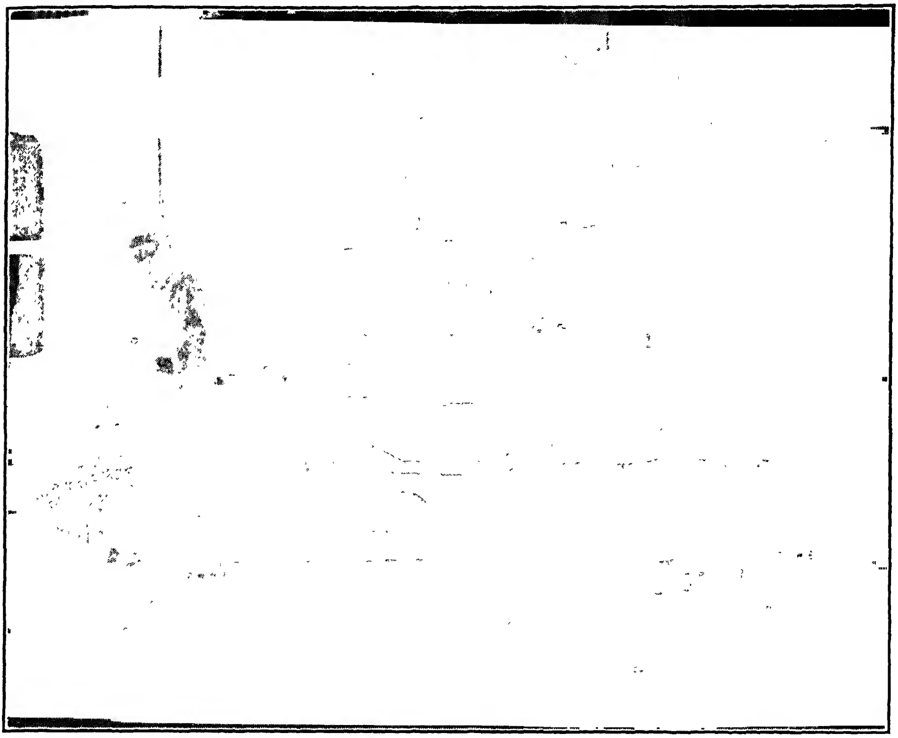

FIG. 10.—CARRIER JUST LEAVING LOADING TANK AND READY TO PLACE DRILL STEEL IN FURNACE.

arm, the carrier moves along the track, down the slope and on to the level part of the track above the furnace; this places the drill-steel bit in the furnace with all parts of it equally exposed to the heat.

The drill carrier now strikes the last carrier so placed and forces it, together with the ten others ahead of it, along the track for a distance equal to the width of one carrier. At this point, the carrier arm releases its grip on the carrier and passing nine of those in the furnace until it comes to the last one, seizes this and pushes out along the track until it reaches the inclined part leading to the quenching tank. The steels

do not move steadily through the furnace, but intermittently, remaining at rest for 30 sec. or longer, then moving a short distance to rest again; thus only four pedestal arms can handle fourteen carriers. At the top of the incline, the grip of the carrier arm is released so that the carriers can run down the incline ahead of it. A braking device prevents a too rapid descent with its consequent jar. As the carrier makes this descent, the drill steel is carried through the door in the end of the furnace to its position over the bubble of water, where it rests for 10 sec. or more with

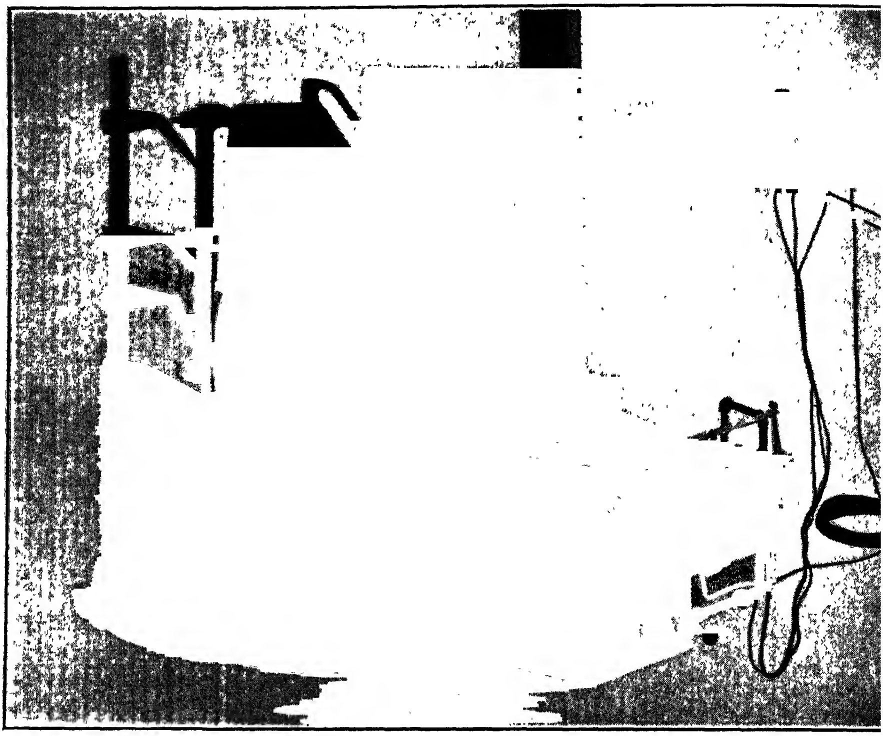

FIG. 11.—DETAILS OF LEVER FOR REGULATING SPEED OF CARRIER.

the water playing on the cutting edges; thus preventing steam bubbles forming. The carrier arm comes along, again grips the carrier, and pushes it with its drill steel, along the slightly sloping track, gradually immersing the drill steel deeper and deeper into the still water in the tank until the maximum depth, about 4 in., is reached.

The track now changes to a sharp upgrade, so that as the carrier is forced along it, the steel is lifted from the water and later forced from the grip of the carrier by steel fingers. The drill steel, now properly hardened, falls to the floor or on to some receptacle provided to receive it,

while the empty carrier continues to the top of the slope and on past the loading table, ready to receive another steel. The steel is picked up by the operator or his helper, examined for the quality of bit and shank and, if correct, placed in its rack.

Output of Heat-treating Machine

The capacity of the heat-treating machine depends on the size of the bits treated. Since it takes longer properly to heat larger steels than

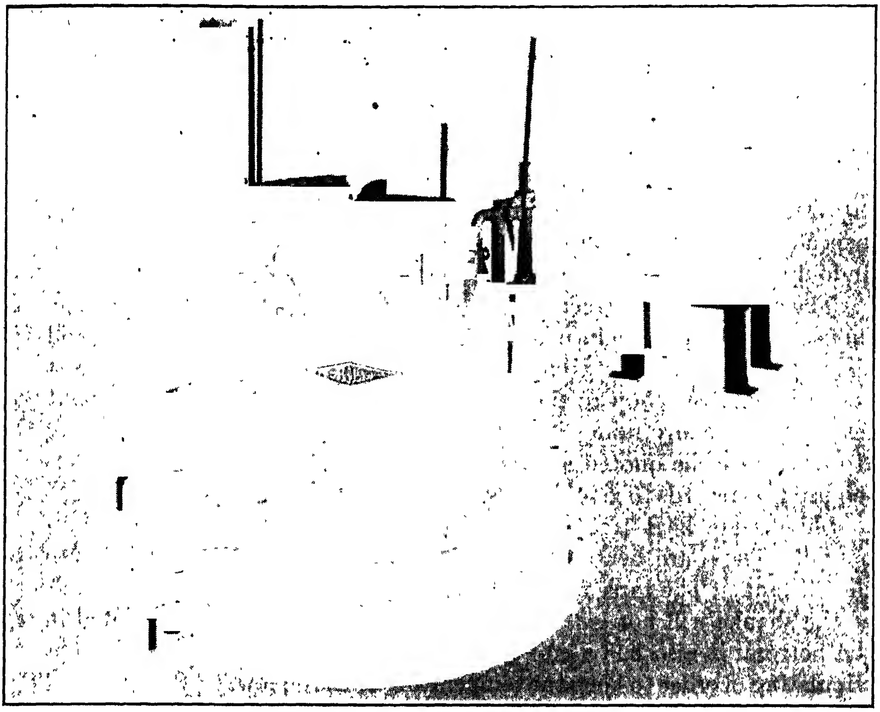

Fig. 12.—Door at end of furnace; drill steel over bubble; steel falling out of carrier.

smaller ones, with the furnace constant, it naturally follows that the machine must be run more slowly when treating bits with large gage and wing thickness. A lever which shifts a speed-changing device is held in various positions corresponding to the size of the bits in the furnace. The speeds for the various sizes are shown in Table 1.

The maximum output is about 112 drills per hour; the minimum, about 56. If the furnace were to be run at a higher heat, a greater speed could be used for the larger bits, though in such a case a larger flow of water should be used in the quenching tank, in order to cool the bits properly. Unless the extra speed is necessary, the

TABLE 1

Lever Position	Steel Size, Inches	Gage, Inches	Time Drill Bit Remains in Furnace	Time Drill Bit Remains in Bubble Seconds	Time Drill Bit Remains in Quenching Tank	Output of Machine Time for Each Bit, Seconds
			Min. Sec.		Min. Sec.	
1	7/8	1¼	5 00	11	0 50	32
2	↑	1½	6 10	12	0 57	37
3		1¾	7 20	13	1 5	43
4		2	8 35	15	1 13	49
5	↓	2¼	9 55	17	1 23	56
6	1¼	2½	11 23	19	1 35	64

machine should be operated as designed. At Franklin, the machine takes care of 750 bits in 8 hr. without trouble. The largest gage handled is 1¾ in., except in rare cases.

Two men operate the machine: the runner places the steels in the carriers, watches the adjustments, etc., while the helper takes them away, inspects them, and racks them. At the slowest speed, one man undoubtedly could do both tasks.

The uniform and correct results obtained from the use of this machine should considerably reduce the amount of steel to be sharpened, and also reduce the transportation costs for drill steels. At Franklin the cost per drill from the shop to the face and back to the shop again is about ten cents; at many mines it is considerably more and at others is less. No figures can be quoted as to a reduction in the number of drill steels required, as records to give that information are not available.

Shanks are heat treated in oil, Houghton's No. 2 quenching compound. These are heated in the sharpener furnaces and plunged into the oil tank. The tank has a screen above the bottom so that the ends of the shanks will not get into any dirt that may have accumulated in the bottom; it also has a coil of pipe through which cold water is kept circulating in order to keep the temperature of the oil low.

TEMPERATURES

For heating drills for sharpening, 1900° F. is allowed. At lower temperatures, the steels do not punch easily and are apt to get too cool for proper sharpening, necessitating another heating. Drills treated in this way have drilled as many as 22 holes (33 ft.) in limestone without resharpening.

For the hardening operation, the drills come out of the heat-treating machine at about 1450° F., though the furnace temperature is about 150° higher than that.

Shanks are heated to about 1600° F., as judged by the blacksmith's eye. Lower temperatures do not give a sufficiently hard shank. At one time it was thought that the proper way to heat the shanks was to hold

the furnace at the temperature desired, so that the steel could not be heated higher than that, but it was found that this caused too much delay. With the furnace full of steels, but kept at 1550°, almost twenty minutes was necessary to bring a cold steel to that temperature. The heat-treating machine is not used for hardening shanks, as the change from water to oil as a quenching medium, and back again, would take take too long. Were enough shanks made up in a day to keep a second heat-treating machine busy, an oil-quenching installation of one of these machines would make an ideal way to harden shanks. At this shop, however, there are not enough shanks made daily to warrant another machine.

Output per Shift

The output per man per shift is about 105 bits or shanks formed and heat treated. In this is included making new drills, cutting old drills to new length, straightening bent drills, opening plugged drills, sorting out the dull drills at the shop, racking the sharp drills and placing the daily requisitions on the trucks. The present load on the shop requires that one drill sharpener be run two shifts and the second sharpener one shift; but more drills could be handled without an increase in the labor employed, should the occasion arise. This of course, would increase the efficiency obtained, to some extent.

Though the methods employed are quite up-to-date, the shop layout is not ideal; the building is too small for the apparatus because of the gradual addition of equipment as need arose or conditions warranted. This crowding made a shop transportation system with tracks impossible and caused the development of the movable tables. These have answered the need very well and have proved more flexible than a track system, unless a connection to an outside system of tracks could have been made.

DISCUSSION

G. H. Gilman, East Boston, Mass. (written discussion).—For many years, the commonly accepted method[1] of heat-treating the cutting end of rock-drill steel, by mine and quarry drill blacksmiths, was to heat the drill bit in an open firebox or coal forge to a dull or cherry red, for a distance back of from ½ to 1½ in., and then quench it by standing it upon its end in a tub, allowing it to remain in the water until it was cool. Sometimes the tub was some distance from the forge so that a perceptible drop in temperature occurred before the drill was placed in the tub. Sometimes there were several inches of sediment in the bottom of the

[1] G. H. Gilman: Heat Treatment of Rock-drill Steel. *Trans.* (1921) 66, 779.

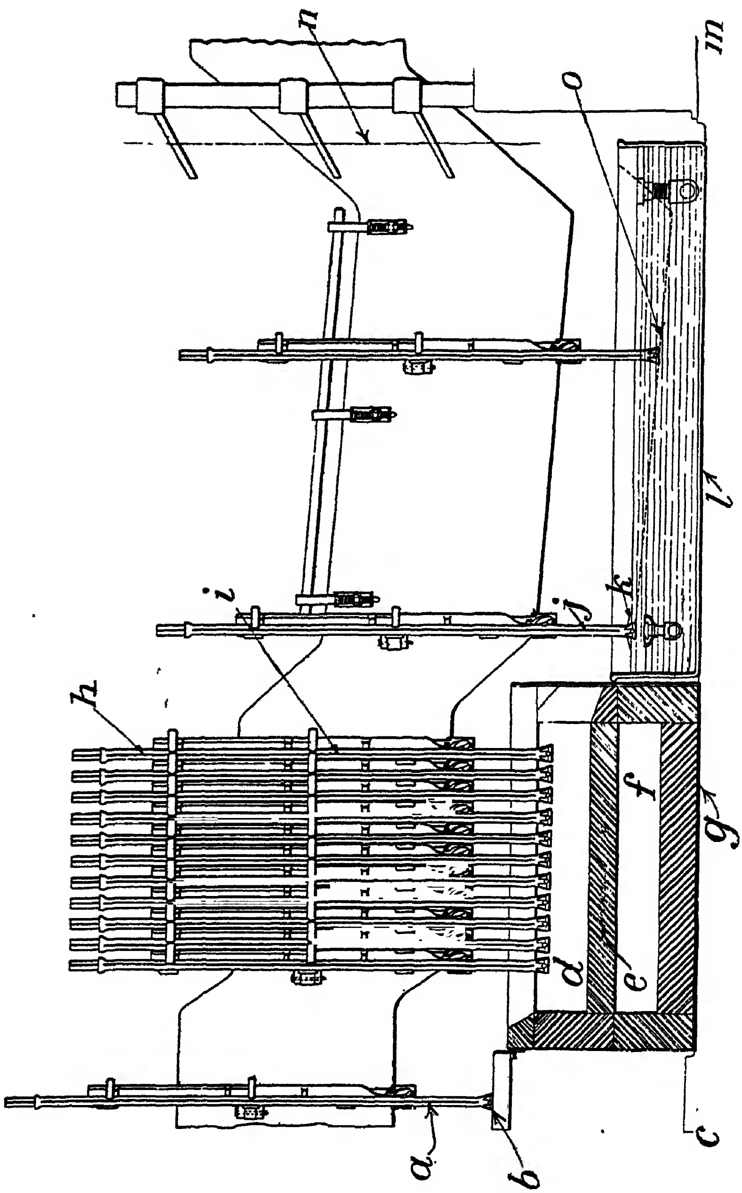

FIG. 13.

tub and sometimes the tub was so small that the quenching media would reach the boiling point before the end of the shift. Some of the more careful smiths drew the temper by cooling the reaming and cutting edges of the bit in the quenching media and then allowing the heat in the bar rearward of the cutting end to run down by conductivity and thus toughen the material rearward of its cutting qualities.

By the first method, there was a tendency for the bit, when submerged 1 in. or more, to harden too far back, which caused brittleness and consequent breakage because of the well defined line of demarkation thus formed between the hard and the soft material. The sediment in the bottom of the tub or steam bubbles prevented even and quick cooling by direct conductivity of the quenching media. The second method was too slow and unreliable when many steels were heat treated.

The two important variables in forging and tempering are time and temperature. As there is a direct relation between the time required to raise the temperature of a mass unit of steel to the upper critical range and the time required to cool it in cold running water, it was possible to so design the machine (Fig. 13) that all thermal units, including furnace temperature, flow of quenching water, height of water bubble, level of water in the tempering tank and the depth to which the drill steel is inserted in the furnace, would be constant. The speed of the driving mechanism is, however, varied by the operator to correspond with the gage diameter of the drill bit, which determines the mass unit of material to be heated and cooled.

The commercial use of the machine is governed entirely by the number of drill steels to be treated in a definite time period. It is claimed

Fig. 13.—Development of Gilman automatic heat-treating machine for rock drill bits.

a. Operator places drill steel in machine at this point.
b. Top surface of loading table serves to gage vertical position of drill steel.
c. Loading table.
d. Heating chamber, in which drill bit is heated in a non-oxidizing atmosphere.
e. Incandescent hearth.
f. Combustion chamber.
g. Furnace.
h. Drill steel is removed from this point in furnace when cutting and reaming edges of drill bit are heated to upper critical range temperature which is gradually blended rearwardly through distance of several inches to neutralize effect of higher forging temperature.
i. But 2 sec. required to transfer drill steel from this point in furnace to water bubble with no perceptible reduction in temperature of drill bit.
j. At this point cutting and reaming qualities only of drill bit are rapidly cooled in a bubble of running water which insures a grain structure of maximum density and maximum hardness.
k. Water bubble.
l. Quenching tank.
m. Ejector.
n. At this point drill steel is automatically ejected from machine ready for use.
o. In following this line drill bit is tempered to insure a gradually blended toughened structure extending from cutting edge through a distance of 2 in. to normal condition of material.

by the manufacturers that the saving effected by its use will more than offset the expenditure of installing it during the first year's operation. This saving is realized directly from the increase in the length of life of the drill steel, which is often 100 per cent. in excess of that shown by the present-day rock-drill steel when heat treated by the hand-and-eye method. This longer life results in: (1) Reduction in the amount of drill steel required for a given amount of work; (2) reduction in cost of resharpening; (3) reduction in cost of transporting the drill steel to and from the smith's shop; (4) increased drilling speed; (5) reduced breakage of drill steel; (6) reduction in power consumption of the drill per foot of drill hole; (7) reduction in upkeep cost of the rock-drilling machine; (8) reduction in the cost of explosive; (9) for any one hole, a reduction in the gage diameter of the subsequent bits used.

Five rock drill bits, all but one of which were improperly heat treated, were subjected to test under a fixed set of working conditions. The work consisted of drilling a 12-in. hole in Barre Vermont granite with a hammer drill that was properly supported and caused to operate under 100 lb. air pressure. The rate of penetration of all drill bits, at the beginning of the test was 8 in. per min. The results of this test are as follows:

Bit No.	Condition of Bit before Test	Drilling Rate at End of Test, Inches per Minute	Condition of Bit at End of Test
1	Proper forging and heat treatment.	7½	Normally worn, cutting edge flattened 1/16 in., reduction in gage diameter 1/64 in.
2	Overheated preparatory to forging, but properly heat treated.	6	Flattened surface at extremities of wings about 3/16 in.; center up to standard.
3	Proper forging, but heated too quickly preparatory to quenching.	4¾	Center was soft; heat did not have time to penetrate core; bit flattened at center; extremities of cutting edge flattened 1/16 in.
4	Proper forging, but underheated preparatory to quenching.	3	Bit mushroomed over; temperature was too low to harden it when quenched at the lower critical range.
5	Proper forging, but overheated preparatory to quenching.	1½	Coarse grains formed that readily disintegrate under hammering action; gage diameter reduced 3/16 in.

Bit No. 2 shows the importance of proper forging preparatory to heat treatment. To forge this drill bit, it was heated in a hot fire and its entire end surface was decarburized in an oxidizing atmosphere. When the bit was subjected to the dollying operation, this decarburized film, which extended over the entire end surface of the bit, was forced to the extremities of the wings so that this part of the bit was decarburized to a much

greater extent than the center. In fact, its center is practically up to standard in carbon content.

The function of the automatic heat-treating machine is to heat treat all drill bits correctly; but it will not make good drill bits from poor steel. If the carbon has been burnt out of the steel before it is put into the heat-treating machine, the machine will not restore it to normal carbon content; but if the bits are properly forged, the machine will heat treat them to the best advantage, provided the cycle of operation as outlined is the proper method of accomplishing the result.

F. B. Foley, Chattanooga, Tenn.—A machine developed in England and South Africa is an interesting use of the magnetic transformation point of steel to automatic control. The steels are placed in muffles which gradually heat the steel up to just below the temperature at which the steel loses its magnetism. In the final heating chamber, in which the steel is heated to just above this point the steel is placed on a carrier with a counterweight and held in place in the magnetic field of a solenoid. The counterbalance pulls out the steel as soon as the magnetism has left the heated part of the steel, which is then ready for quenching. There is no automatic method of quenching. This withdrawal method is particularly applicable to drill steel, because mine drill steel has but one transformation temperature—at which magnetism is lost and recrystallization has been attained and above which steel, when quenched, will harden. Some of the bits shown me were particularly good.

One of its features is that below the temperatures at which steel hardens there is very little scaling and practically no decarburization, so the steel is brought up to hardening temperature in an atmosphere and temperature at which practically no decarburization can take place. Then it is ready to jump through the transformation temperature very quickly when exposed to the temperature at which it might scale and decarburize.

B. F. Tillson, Franklin, N. J.—This South African furnace, known as the O'Donovan, does not directly mechanize the heat treatment. The steels must be moved forward in these muffles from zones of one temperature to zones of increasing temperature, and they must be withdrawn and passed, in individual cylindrical muffles, to be within the magnetic influence so that they may be withdrawn by the counterweight when they reach the critical temperature. That is one reason why it may not be as widely adopted as some method of heat-treating, handling and quenching drill steels that is mechanized as a whole and to a greater degree. In addition, it is large and the cost of installation is considerable.

If you can control the temperature of a furnace so it cannot exceed a satisfactory temperature, it is not necessary to withdraw the steel the exact second at which critical temperature is reached. Is there not a lag

in the performance of any electromagnetic method because of the rapid reduction in strength of the magnetic field as the distance of the de-magnetized body from the source of that field increases and because the difficulty of placing magnets close enough to the heated drill steels to avoid the influence of variability and magnetic effects?

The point was made by the producers in South Africa that the appearance indicates an unusually superior quality of drill bit. Possibly after $\frac{1}{32}$ in. had been worn away, a hard rib would develop, that was considered very unusual and showed the high quality of heat-treatment of the steel. It is not unusual. At Franklin, 17 or 18 years ago with manual methods of heat treatment of drill steel, we produced many steels that showed the same effect, which frankly, we did not consider a sign of merit and abandoned that method. We decided that the underlying hard rib, if you may call it so, indicated that the surface of that drill bit had not been heat-treated, otherwise it would have been as hard as that rib showing through.

G. H. Gilman.—A 3-hp. electric motor mounted immediately below the leading table drives the blower that provides the air for the burner. Part of the air supplied by the blower is conducted to a curved perforated pipe laid on the top of the furnace and immediately rearward of the arc-shaped opening, into which the bit end of the drill steel is inserted. This air blast is directed across the top of the opening and prevents egress of the products of combustion and limits the distance through which the heat travels up the drill steel. The furnace temperature, which is maintained approximately 150° higher than the upper critical range of the steel, is indicated by an electric pyrometer, the thermocouple of which is inserted in the top center of the furnace immediately forward of the slot through which the drill steels pass. Any variation in the temperature of the furnace causes a small motor to open or close the fuel-oil control valve. A thermocouple inserted through the wall of the furnace, at its outlet end and adjacent the heated drill bit just before it is transferred to the water bubble, provides a means of determining the exact temperature of the heated drill bit, in degrees Fahrenheit.

The machine provides mechanism for coördinating the function of each feature. One man places the unhardened drill steel in the jaws of one of the carriers. The machine automatically transports it from the the loading table into the furnace, where its temperature is progressively raised until the correct heat for quenching is attained. Then it is quickly transferred into the quenching tank and, for a few seconds, the cutting and reaming edges of the bit are subjected to the impinging action of a jet of cold water; this imparts to them maximum hardness and density (the desired wear-resisting qualities). Then, a core of the required toughness is secured by slowly submerging the bit, as it is caused to travel

throughout the length of the quenching tank. When thus cooled, it is raised from the quenching tank and ejected from the machine ready for use.

W. R. Wright, Chicago, Ill.—What is the total operating cost for each machine?

G. H. Gilman.—One man can operate a heat-treating machine; the electric power consumption averages 2 hp. Although a 3 hp. motor is installed the equivalent of 3 hp. in electrical energy is not used. The water cost, of course, will vary with the conditions. The fuel consumption of the furnace, when oil is used, approximates 2 gal. of fuel oil per hour.

W. R. Wright.—What is the cost of the machine installed?

G. H. Gilman.—The initial installation fee is $1800, and the rental fee, $60 per month in the United States, Canada and Mexico.

R. M. Raymond, New York, N. Y.—What is the number of drills it will handle, at a moderate speed?

G. H. Gilman.—The maximum capacity is one drill steel every 30 sec. or 1000 drill steels per eight-hour shift.

R. R. Van Valkenburgh, Wharton, N. J.—We use Ingersoll-Rand No. 50 drill-sharpening machines, but depend for heat treatment on the old fashioned method. We believe that the place for sharpening is underground, as near the actual mining operations as possible. The average number of pieces handled in our shop, during 1925, was 665 per day. This includes not only resharpening but the making of new drills and the reworking of drill shanks. This average indicates a cost of 7 c. per piece at the shop; and as the handling charge is 3 c., the cost of a drill at the working face becomes 10 c.

We get few complaints about poor steel. Most of our mining is on a piece work basis, or the man who receives a poor run of drills takes up the matter personally and forcibly with the sharpening boss, which we find gives a mutual benefit.

Without exception, we find soft drill bits are caused by too much heat or too long a heat at the forging operation. Too long a heat is as disastrous as too great a heat. The heater must be impressed with the fact that a minimum number of pieces, commensurate with the sharpening speed, should be maintained in the forge and that these should be carefully rolled through the heat. When a soft bit, or one with a soft center or soft wings returns to the shop, the bit is cut off and a new one forged on. There seems to be very little practical difference in the hardening practice for steel ranging in carbon content from 0.7 to 0.85 per cent. Steels with the lower content seem better able to resist fatigue.

R. M. Raymond.—About what depth of hole will the fresh steel cut?

R. R. Van Valkenburgh.—The difference in resistance or hardness between the ore and rock is very marked and the drill steel is designed to cut the rock, which is the harder. By allowing a change in bit gage of ¼ in. for the first and second drills and ⅛ in. thereafter, it is possible to arrange an 18-in. run per each drop in gage. In rock, the entire gage is used in this run; in ore, the drill may be used two or three times, indicating a run of 36 and 54 inches.

J. B. Stewart.—Ten years ago, I asked the Ingersoll-Rand Co. and the Sullivan Machinery Co., to send me the most skilled blacksmiths they had for the purpose of conducting a test on extremely hard quartzite on which no drill did more than 4½ in. per bit. All of the bits tempered by the Sullivan Machinery man after the first day's experience, averaged 18 in. or better. Although he used no magnet, I found, by using a magnet, that he was tempering at the critical temperature on the rising heat.

Three blacksmiths had not been able to supply enough steel, but when they learned to temper by means of the magnet test, one blacksmith was able to keep all of the machines in steel and do all of the blacksmith work around the place. An accurate record of the work done showed an average of 18 in. per bit, and some bits hung had drilled 3 ft.

Top Slicing in Old Fills at El Bordo Mine, Mexico*

By R. J. Mechin,† Pachuca, Mexico

(New York Meeting, February, 1926)

DISCUSSION

R. M. Raymond, New York, N. Y.—The filling and drawing down of the overhead material was done at considerable depth, which is not the usual method in which it starts at the surface.

R. J. Mechin.—It is a little more than 1000 ft. at El Bordo. The vein above had been caved practically to the surface and some of the ore had been excavated, using underhand and overhand square-set methods. This was far enough away from the shaft to prevent any bad effects. When the sill floor was taken out, to insure the old cave material following down well, we put chutes in the square sets for several weeks until we were sure that the material from above was moving down; on each level above, several crosscuts were driven into the old fill by which we could see the material coming down and thus be sure it was following the first. We had to dynamite it very little because it was dry and dusty. The only trouble was that timber starting from the old square sets would get across the chute and hang up.

R. M. Raymond.—How many floors did you operate at once?

R. J. Mechin.—We would operate one floor through the whole length and prepare for the second.

F. W. Sperr, Houghton, Mich.—Could you have worked it in steps?

R. J. Mechin.—Yes; but working it all on one floor was much better for ventilation. The passage between steps would tend to become heavy and blocked and, as the passages from the mining floor down are usually filled with ore, there would be no way for the air to pass up. By means of doors and fans, air was driven to the extreme end of the orebody through the stope and up the other end; but it could have been worked in steps.

F. W. Sperr.—Did the following-up material ever become so heavy that you could see weight on the caps before you had to take out the posts?

* *Trans.* (1925) **72**, 139.

† Formerly mine superintendent at El Bordo.

R. J. MECHIN.—Yes. The weight was there almost by the time the post was capped and blocked. Only about one half of the posts that were left after the floor was down had to be dynamited. But on the foot-wall side, the original drift, very heavy timber, at least 10 to 13-in. posts, was used for we tried to keep that part in good condition. The vein was so narrow that it could be worked very rapidly. The widest point was 36 ft.

F. W. SPERR.—Why was the mine worked by the underhand square set method?

R. J. MECHIN.—It was a good method. Overhead square setting in this sort of ground would need splicing, but underhand square setting required no splicing. The maximum was 36 ft. with a depth of 82 ft.

R. M. RAYMOND.—That was fill and the same method was applied there?

R. J. MECHIN.—Yes.

F. W. SPERR.—The hanging wall was strong and did not break?

R. J. MECHIN.—No. Ordinarily, it was not necessary to put down a floor on each slice, but this fill was so dry and fine that it had to be done. We tried to do without it at Santa Gertrudis, but tons came through a small opening. The purpose is mainly to keep the roof overhead.

W. S. PALMER, Reno, Nev.—Something similar is being done in the United Comstock mines.

G. A. PACKARD, Boston, Mass.—You say that all of this material was handled with wheelbarrows; could the scraper be used there?

R. J. MECHIN.—It could have been with a wider orebody and more efficient labor, but with Mexican labor, the fewer mechanical contrivances the better. The maximum distance traveled with the wheelbarrows was 45 to 50 ft.

G. A. PACKARD, Boston, Mass.—In the Grass Valley mines, the distances covered with the scraper are short; not more than 60 or 70 ft. The Christmas mine, in Arizona, with a flat stope has substituted the scraper for wheelbarrows. On the second day, two men with the scraper handled 65 tons, whereas five men with wheelbarrows had been required to handle that in a day. The distance varied from 40 to 90 ft. The stopes were very flat and a heavy scraper was used. They were paying $3.25 for that particular kind of work.

C. M. HAIGHT, Franklin, N. J.—At Franklin, it is difficult to use scrapers in the top slices because of timbers; besides the ore breaks in too large chunks. Where the ore must be carried more than 10 or 15 ft. to

a chute, small tram cars, a little larger than wheelbarrows, called buggies, are used. They are about 24 in. above the rail, hold about 2 or 3 wheelbarrow loads, and are pushed on the rails to the ore chute.

In most of the work we try to get the raises from the haulage level close enough together to avoid transporting the ore in the top slice itself. We put the raises in the center of the pillars when the pillars are not more than 35 or 40 ft. wide, about 20 ft. center to center. That eliminates all wheelbarrow work or tramming. Of course, in many mines so many raises would not be practical.

Liquid Oxygen as an Explosive

By Frederick W. O'Neil* and Herman Van Fleet,† New York, N. Y.

(New York Meeting, February, 1926)

Scope of This Report

The object of this paper is to describe the present status and possibilities of liquid oxygen as an explosive based upon the investigations, research and practical work of the Ingersoll-Rand Co., and of the Air Reduction Co., from early in 1922 to date. The following work was done.

1. A study of the history and physical theory of L. O. X. as an explosive. (No attempt will be made in this paper to give the history or theory, as this has already been covered by the literature on the subject.)

2. An investigation of the use and the practice with L. O. X. in the iron mines of Lorraine where L. O. X. has its largest application in Europe. Although L. O. X. is extensively employed in the iron mines of Lorraine, many of the mines using it exclusively, and blasting with it a total of approximately three million tons of iron ore per year, the fact that the mining practice in Lorraine[1] is to drill comparatively shallow holes and blast only a few holes at a time, renders the data obtained there of very little value in estimating the possibility of replacing dynamite with L. O. X. in American mining methods. In fact, in the iron mines in Lorraine, L. O. X. replaced black powder, or its equivalent.

3. Laboratory research work on properties of absorbent materials.

4. Work in the Bureau of Mines explosive laboratory at Pittsburgh, Pa., on the properties of L. O. X. cartridges.

5. Practical work underground with L. O. X. cartridges in the Witherbee-Sherman iron mines at Mineville, N. Y.

6. Practical work in the quarries of the Calcite Quarry Co., Myerstown, Pa.

7. Practical blasting in several quarries and open-pit mines.

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* Frederick W. O'Neil, chief engineer, Ingersoll-Rand Co.

† Herman Van Fleet, vice president, Air Reduction Co.

[1] For a full description of the practice in the use of L. O. X. in the iron mines in Lorraine see "l'Emploi de l'Explosif a l'Oxygene Liquide, dans les Mines de Fer de Lorraine." Assn. Miniere d'Alsace et de Lorraine (Dec. 1, 1922).

Production of Liquid Oxygen

Reliable apparatus for the development of liquid oxygen has been fully developed and perfected. Two general methods are used: (*a*) The so-called "Linde" process where refrigeration is obtained by the Joule-Thompson effect of free expansion through a nozzle. This method involves compressing the air to a pressure of 3000 to 3500 lb. per square inch.

(*b*) The Claude system developed by the Compagnie L'Air Liquide in France, and the Air Reduction Co. in the United States. In this process, refrigeration is obtained by expanding a part of the air in an expansion engine. Much lower pressures, and correspondingly lower powers are needed; the pressure required depending on the size of the plant.

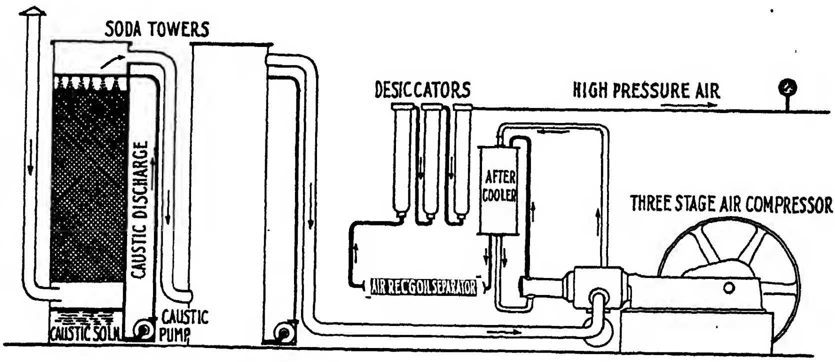


Fig. 1.—Flow sheet of a liquid-oxygen plant, Claude system.

A plant producing 75 liters of liquid oxygen per hour requires that the air be compressed to 900 lb. The method of producing liquid oxygen consists in first producing liquid air, which is then subjected to a rectification or fractional distillation process that separates the oxygen from the nitrogen by utilizing the difference in their boiling points. (Figs. 1 and 2 are plans of the Claude system.) The plant consists essentially of: Soda towers, in which the atmospheric air is treated with a caustic soda solution to remove dust and CO_2; a compressor for compressing the air; aftercoolers for cooling the air; separators for removing moisture; desiccators for treating the compressed air with caustic soda and calcium chloride for removing remaining traces of water vapor, CO_2 and oil; and an oxygen column complete with liquefier, heat exchanger, and expansion engine for the production and rectification of the liquid air into liquid oxygen.

Table 1 gives the cost of producing a liquid oxygen in plant making 75 liters per hour.

TABLE 1.—*Cost of Producing Liquid Oxygen with 75-liter Plant*

24-hr. Operation; 28-day Month

Labor: 1 superintendent @ $200.00 mo	$ 200.00
2 operators @ 175.00	350.00
1 laborer @ 125.00	125.00
Supplies: Caustic, 3050 lb. @ $0.04	122.00
Oil and miscellaneous	30.00
Power: 87,500 kw.-hr. @ $0.01 (175 hp.)	875.00
Interest and depreciation: Upkeep and repairs	1,210.00
	$2,912.00
Production: 672 hr. @ 75 liters per hr	50,400 liters
	127,000 lb.
Cost per liter	$0.0577
Cost per lb	0.0229

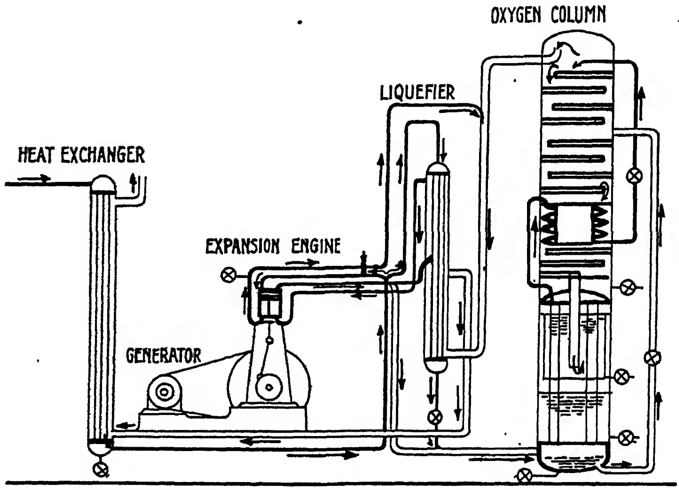


FIG. 2.—FLOW SHEET 2, LIQUID-OXYGEN PLANT, CLAUDE SYSTEM.

CARTRIDGE REQUIREMENTS

Theoretically, the oxidation of pure carbon to CO_2 requires 2.67 units of oxygen by weight for each unit of carbon, and 1.33 units for oxidation to CO. As commercial carbons are not chemically pure, and as the cartridge wrapper is cellulose and not carbon, we have assumed values of 2.5 and 1.25 as the amount of oxygen needed to oxidize the cartridge material to CO_2 and CO respectively. The ideal requirements for a cartridge are as follows.

(*a*) Maximum density in order to provide the maximum of explosive in a given volume.

(*b*) Ability to absorb liquid oxygen in excess of its requirements for complete combustion in order to provide excess oxygen for evaporation before detonation.

(*c*) Physical properties of filling material and wrapper which make the soaked cartridge mechanically strong so that it can be handled without breakage. This applies particularly to the wrapper which must not be too brittle at the low temperature of the liquid.

(*d*) The price of the filling material must be low enough for commercial use.

(*e*) The soaked cartridge must be firm and dry. With some filling materials, especially when lightly packed, oxygen liquid readily squeezes out of the cartridges making them difficult to handle.

It should be noted in general that a material that gives the maximum requirements of density tends to give the minimum capacity for absorption, and, as will be afterwards shown, a commercial cartridge must be a compromise between these two requirements.

Absorbent Materials

Approximately 400 varieties of carbon blacks, lampblacks, hydrocarbons and various mixtures were investigated. Many of these materials were rejected in the laboratory without practical tests, because of high cost, low densities or low absorption qualities. Several varieties of carbon blacks and lampblacks of satisfactory cost (from $0.06 to $0.09

Table 2.—*Ratios of Absorption (Grams Oxygen Absorbed per Gram of Filling Material) Obtained in 1¼ by 8-in. Cartridges*[a] *with Various Filling Materials*

Sample No.	Description	Dry Weight of Cartridge, G.	Ratio	Remarks
1	Wood-pulp carbon	52.5	2.58	
9	Wood charcoal	54.0	1.55	
20	Activated charcoal	56.0	0.90	
23	Granulated cork	18.0	4.66	Bad mechanical properties
103	Lampblack No. 1	20.0	7.10	Price prohibitive
31	Lampblack No. 2	19.0	6.36	Price prohibitive
43	Lampblack No. 3	18.0	5.30	
40	Lampblack No. 4	50.0	1.60	
55	Bone black	60.0	0.50	
96	Acetylene black	30.0	4.00	
29	Carbon black No. 1	33.0	4.21	
29*a*	Carbon black No. 2	33.0	5.20	
39	Charred balsa wood	3.0	17.00	Density too low
189	Brown lampblack	17.0	8.80	Not available in quantity
184	Carbene	34.0	3.60	
154	French cartridge material	44.0	4.20	
95	Pachuca black	32.0	4.00	
	German cartridge material	54.0	2.60	

[a] Dry weights given above include 5 g. of wrapper. In each cartridge the material was packed just tight enough to give a firm cartridge which could be handled.

per pound) and of satisfactory density and absorption qualities[2] were selected.

Table 2 shows the variation in density and absorption qualities of some of the most promising materials investigated. From this table it is easy to see why the lampblack and carbon black represented by samples Nos. 43 and 29*a* respectively, were finally chosen as the best absorbents, and our further laboratory testing and practical use was confined to these materials. These absorbents are procurable at reasonable prices and are available in large quantities.

Wrappers

The requirements for a cartridge wrapper are as follows: (*a*) The material should be combustible; (*b*) it must possess the property of allowing liquid oxygen to permeate it readily; and (*c*) it must not be brittle or fragile in soaking, to permit the soaked cartridges to be handled without breakage. It must tend to retain the liquid oxygen, and at the same time allow sufficient evaporation to prevent rupture of the cartridges by expansion of the gas.

Table 3 shows the properties of the most promising wrapping materials investigated. The material finally selected was pure rag stock paper varying in thickness from 0.007 to 0.010 in. All papers which are filled or sized are unsuitable owing to brittleness when soaked.

Table 3.—*Effect of Wrapper on Evaporation of Oxygen from 1¼ by 8 in. Cartridge*

Test No.	Wrapper	Dry Weight of Cartridge, G.	Time to CO_2 1 Min.	Remarks
82	Heavy brown wrapping paper.	27	6	
83	Coarse filter paper...........	32	8	Mechanically weak
84	Black hard fibre paper........	30	9	Brittle when soaked
86	Gray hard-fibre rag, 0.01.....	26	11	Brittle when soaked
161	Gray crepe paper............	30	6	
163	Red rag stock 0.01 caliper....	32	5½	
165	Red rag stock 0.007 caliper....	30	7	This paper adopted

Considerable difficulty was experienced in making a suitable cartridge because they cannot be made with paper which has been gummed or glued, as this makes them impervious and brittle. On the other hand, a cartridge wrapped spirally without gumming will allow the filling material to sift out. We finally developed a method of making a cartridge of a piece of paper of rhomboid shape rolled on a mandrel. A gumming

[2] For information on the production of carbon blacks and lamp blacks, see "Carbon Black—Its Manufacture, Properties and Uses" by R. O. Neal and J. St. J. Perrott, U. S. Bur. of Mines *Bull.* 192 (1922), 95 pp.

machine gums one edge of this paper so that when rolled this gummed edge encircles the cartridge spirally. This machine crimps one end of the cartridge, and the other end is crimped by hand after it is filled. The ends after filling are immersed in paraffin for about ¼ in., and this successfully prevents sifting.

We found that practically all of the cartridges made in Lorraine were filled by hand.

Early in our work we discovered that we could not make cartridges successfully by filling them by hand, because of the great variation in density, not only of the different cartridges, but within a given cartridge. We, therefore, developed a machine for filling the cartridge. This consists essentially of a hopper containing the cartridge material. The hopper contains a spiral screw that causes the filling material to exude through a nozzle. The cartridge is placed upon a weighted platform underneath the filling nozzle. This weighted platform is arranged so as to oppose any desired resistance to the flow of material from the nozzle. In this way, the cartridges are filled to uniform density. We have no difficulty in keeping the density desired within a variation of 2 or 3 per cent.

The term "density," as used in this paper, indicates mean apparent density obtained by dividing the gross weight of the cartridge in grams by the gross volume in cubic centimeters.

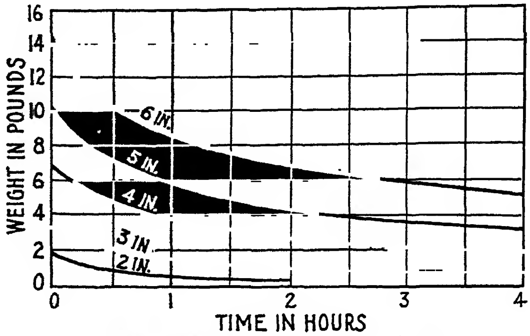
Table 4 shows the cost of unsoaked cartridges based on black at $0.085 per pound.

TABLE 4.—*Cost of Cartridges*

7 in. by 20-in. cartridges @ 8 lb. each:	
Carbon 8 lb. @ $0.085	$0.680
Bag heavy duck	0.335
Labor	0.075
Supervision, power and overhead	0.100
Cost per lb. $0.15	$1.190 each
4½ in. by 18-in. cartridges @ 3 lb. each:	
Carbon 3 lb. @ $0.085	$0.255
Bag medium duck	0.100
Labor	0.030
Supervision, power and overhead	0.035
Cost per lb. $0.14	$0.420 each
1¼ in. by 12-in. cartridges @ 6 cartridges per lb.	
Carbon ⅙ lb. @ $0.085	$0.015
Wrapper rag paper 0.007 in	0.010
Labor	0.010
Supervision, power and overhead	0.005
Cost per lb. $0.24	$0.040 each

Influence of Size and Density on Life of Cartridges

Fig. 3 shows the relation between weight of soaked cartridges and time exposed in the open air after soaking, for large cartridges. Fig. 4 shows the same for small cartridges.

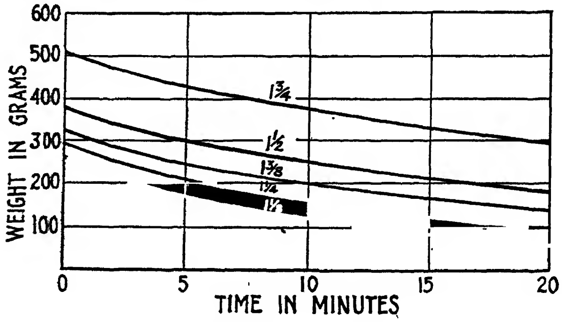


Diam., In.	Dry Weight, Lb.	CO_2 Wt., Lb.	CO Wt., Lb.
6	3.500	12.24	7.88
5	2.375	8.32	3.35
4	1.625	5.69	3.66
3	0.843	2.95	1.90
2	0.469	1.64	1.06

Carbon in cotton bag.

Fig. 3.—Typical time-weight curves for 12-in. cartridge of large diameter.

Fig. 5 shows the time in minutes taken for the evaporation of excess oxygen in the cartridge down to the amount necessary for combustion to CO_2 for cartridges of different densities but of the same composition.



Diam.	Dry Wt., G.	CO_2 Wt.	CO Wt.
1¾	112	392	252
1½	83	290	187
1⅜	65	230	148
1¼	57	200	128
1⅛	46	161	104

Carbon in paper wrapper.

Fig. 4.—Typical time evaporation curves for 10-in. cartridges.

Offhand, this graph would indicate that the low-density cartridges are desirable as they give a longer life; however, as the amount of carbon present is smaller, their strength per unit of volume is decreased. There-

fore, the selection of the proper density depends upon a compromise between requirements for long life (low density) and explosive strength (short life and high density). It is noteworthy that this table shows that when the cartridge is packed to a low density it is unable to hold the liquid oxygen, and some of the liquid runs out of the cartridge when it is removed from the soaking bath.

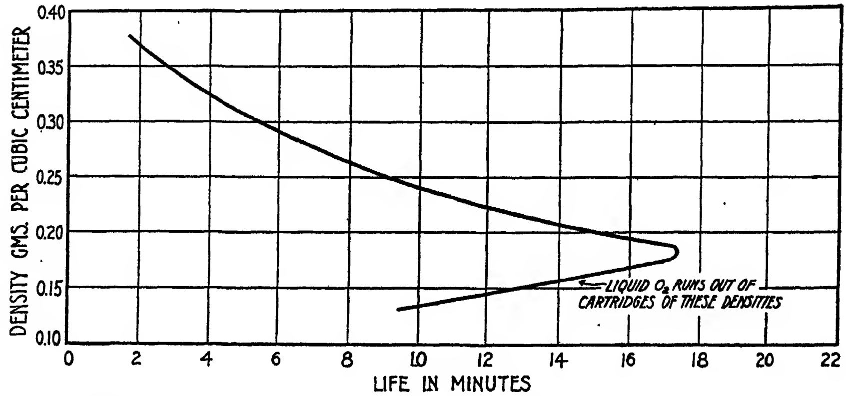


FIG. 5.—VARIATION OF CARTRIDGE LIFE WITH DENSITY; 1½ BY 10-IN. CARBON IN PAPER WRAPPER.

CARBENE

It will be noted that in Table 2 reference is made to carbene. This material is a polymer of acetylene $C_n H_n$ that has received a good deal of attention, especially in Europe, as an absorbent superior to lampblack. It is made by admitting acetylene gas into a closed retort maintained at a temperature of approximately 290° C. Originally it was claimed that cuprous oxide was a necessary catalyst for its production. We had no difficulty in making carbene in commercial quantities and found that the use of cuprous oxide was not necessary, as a little carbene apparently serves as its own catalyst. We see no hope of its use as an L. O. X. absorbent material because first, based on American costs of carbide its cost is prohibitive, and, second, our tests show it is manifestly inferior to lampblack.

STORAGE AND TRANSPORTATION OF LIQUID OXYGEN

Liquid oxygen is stored and transported in Dewar flasks or vacuum bottles. Fig. 6 shows the principles of construction. These vessels consisted essentially of an inner and outer shell with a vacuum space between them and containing a compartment in which is placed a small amount of activated charcoal. The outer vessel is furnished with a lead tube which is connected to an air pump to exhaust the vacuum space. When the required vacuum is attained, the lead pipe is pinched off and

soldered over. At the low temperature which obtains when the flask is filled with liquid oxygen, the activated charcoal absorbs the residual air in the vacuum space. The whole flask is then surrounded by a sheet-steel case; the space between the flask proper and the outside container is filled with excelsior, or similar material.

These flasks are very efficient. Fig. 7 shows the results of tests on the loss from evaporation from 15 and 5-liter containers at rest. Data was also acquired as to the amount of evaporation taking place from containers during shipment combining 3 to 4-day railroad and motor truck transportation of several hundred miles. The average loss in transit for eighty 15-liter containers was 10 per cent. per 24 hr. This was obtained from flasks which at rest had a loss from evaporation of 7.6 per cent. per 24 hr.

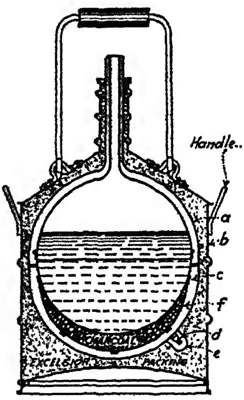
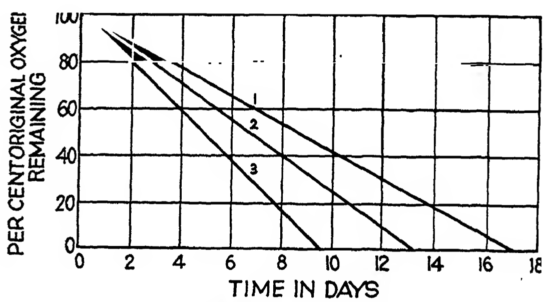


FIG. 6.—TRANSPORT CONTAINER.



No.	Wt. Empty, Lb.	Wt. Full, Lb.	Capacity, Lb.
1	30	68.5	38.5
2	35	70.5	35.5
3	14	28.0	14.0

Per cent. loss per 24 hr.

FIG. 7.—EVAPORATION FROM OXYGEN CONTAINERS AT REST.

We believe it entirely feasible in liquid-oxygen plants to store the liquid in double receivers large enough to hold 24 or 28 hrs.' supply. Although we have no data as to the amount of evaporation which would take place from vessels of large capacity, we are of the opinion that owing to the large ratio between the volume and radiating surface as compared to small flasks, vacuum insulation of such receivers would probably be unnecessary, and hair felt or mineral wool could be used instead. These receivers should be made of copper or copper alloys, as steel is unsuitable for storing liquid oxygen because it is liable to develop fine cracks at low temperatures.

MAINTENANCE OF TRANSPORTATION CONTAINERS

Our experience has not been extended long enough to give very satisfactory data on the maintenance of vacuum equipment. Of fifty-five

15-liter containers, at the end of one year 15 had to be reevacuated. The loss of efficiency in containers is due less often to loss of vacuum than to the fact that the inner and outer shells of the vacuum space are brought into permanent contact by displacement from rough handling. The reevacuation of the vessels is not hard, and can be readily carried out at the mine. A suitable vacuum gage and a small 2-stage oil-immersed vacuum pump are all the equipment required. A vacuum of $\frac{1}{20}$ mm. of mercury is sufficient.

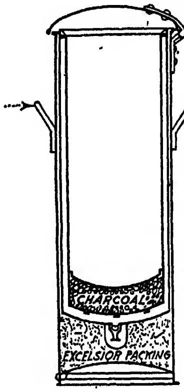


FIG. 8.

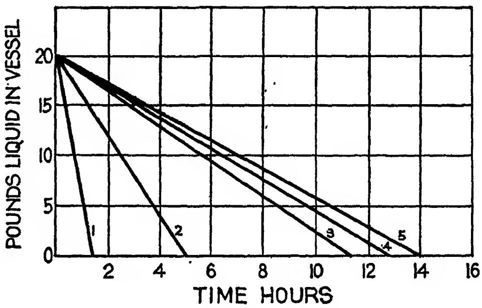


FIG. 9.

FIG. 8.—SOAKING CONTAINER.
FIG. 9.—EVAPORATION TESTS OF SOAKING VESSELS: 1, UNINSULATED COPPER CAN; 2, CORK-BLOCK INSULATION; 3, HAIR-FELT INSULATION AFTER 2 MO. USE; 4, HAIR-FELT INSULATION, NEW; 5, VACUUM INSULATION. ALL VESSELS HAD APPROXIMATELY THE SAME CAPACITY.

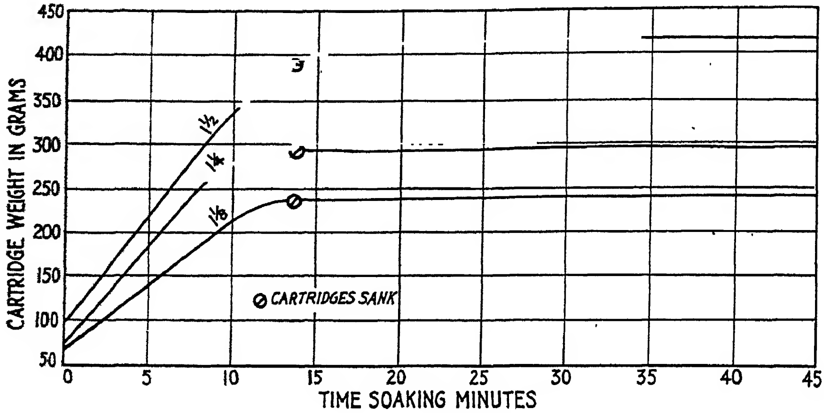
SOAKING CARTRIDGES

The cartridges are soaked in straight-sided Dewar flasks. These soaking vessels (Fig. 8) have the same principle of construction as the transportation containers. In order to facilitate handling, the cartridges are placed in copper-wire baskets which fit easily inside the container. Before starting to use a container, a small amount of oxygen should be placed in it to chill it before soaking the cartridges. Fig. 9 shows the results of tests on the evaporation of liquid from soaking vessels of different constructions. Curve 5 is for the vacuum vessel shown in Fig. 8. The high efficiency of the hair-felt insulated container gave rise to the use of the dry soaker described later (Fig. 12).

TIME REQUIRED TO SOAK CARTRIDGES

The cartridges are placed in a copper-wire basket and then placed in a soaking container, and the liquid oxygen poured in. Fig. 10 shows the

results of tests, giving the time required to soak cartridges of various sizes. It will be noted that when the cartridges sink, they are practically saturated, and that the time required is from 15 to 18 min. A 5-in. cartridge requires about 30 min. for complete soaking. Fig. 11 shows the method of pouring liquid oxygen on the cartridges.



Size, In.	Dry Wt., G.
$1\frac{1}{8}$ by 12	60
$1\frac{1}{4}$ by 12	65
$1\frac{1}{2}$ by 12	101

FIG. 10.—TIME REQUIRED TO SOAK CARBON CARTRIDGES, BEGINNING WITH POURING OXYGEN ON THE CARTRIDGES.

TRANSPORTING THE SOAKED CARTRIDGES

Because the soaking containers shown in Fig. 8 are rather awkward to handle and are liable to be damaged in handling, and because, as disclosed by Fig. 9, a hair-felt insulation has a relatively high efficiency, we developed what we called a "dry" carrier (Fig. 12). This consists of a double-wall vessel with about $1\frac{1}{2}$ in. of hair-felt insulation between the walls, the inner vessel being made of copper and the outer one of steel. After the soaked cartridges are removed in the basket from the soaking container, they are placed in the dry carrier, in the bottom of which a little liquid oxygen has already been placed. The cartridges then tend to keep saturated by capillarity from the oxygen in the bottom of the vessel.

Tests to determine the ability of a cartridge to soak up liquid oxygen by capillarity are shown in Fig. 13, which gives the effect of standing a *dry* unsoaked cartridge on end in one of these dry carriers with a little liquid in the bottom, the liquid being maintained at a constant level. The test shows that the dry cartridge is able by capillarity alone to absorb enough oxygen to reach a point between its CO and CO_2 requirements.

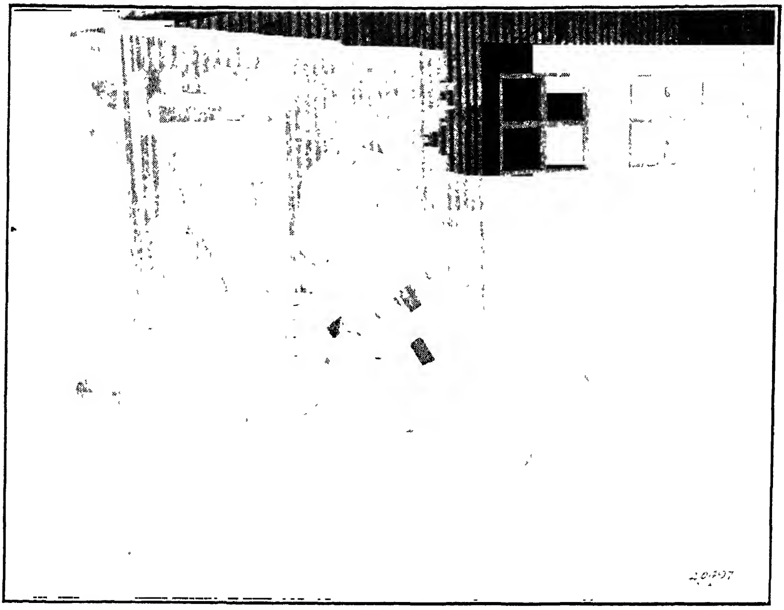


FIG. 11.—POURING OXYGEN ON CARTRIDGES AT CENTRAL SOAKING STATION; SHOWS CARTRIDGES IN BASKET, A SOAKING CONTAINER AND DRY CARRIERS.

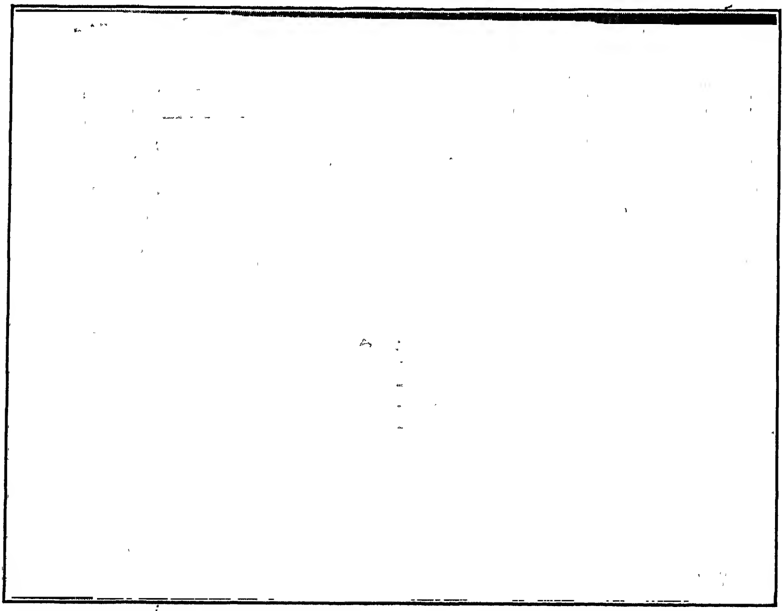


FIG. 12.—TRANSFERRING SOAKED CARTRIDGES TO DRY CARRIER AT CENTRAL SOAKING STATION.

Fig. 14 shows the evaporation that takes place from soaked cartridges that have been placed in a dry carrier with a little liquid in the bottom. This proves that it is entirely practical to keep cartridges in this way 1½ hr. after soaking, and before use. The results shown in Fig. 14

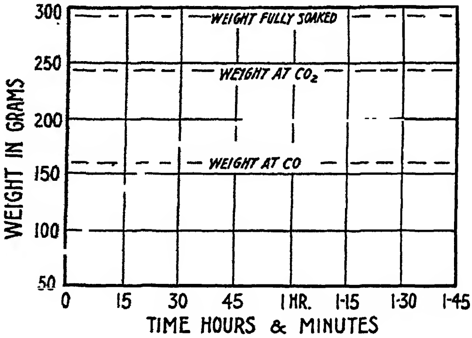
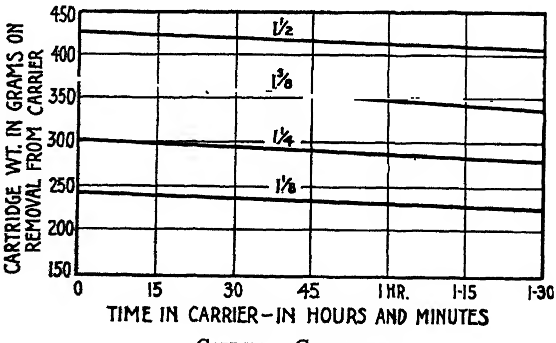


Fig. 13.—Capillarity test 1¼ by 12-in. cartridge. Dry cartridge on end in 1-in. liquid; liquid maintained at constant depth; dry weight of carbon cartridge, 69 g.

were obtained with cartridges of rather low density, and with the carrier only two-thirds full. With cartridges of higher density, such as are generally used, and with the carrier filled with cartridges, the loss of oxygen shown would be reduced by about one-half.



Carbon Size, In.	Cartridges Dry Wt., G.
1⅛ by 12	60
1¼ by 12	65
1⅜ by 12	85
1½ by 12	101

Fig. 14.—Oxygen loss in dry carrier.

As a result of the development of this procedure the vacuum soaking-containers need not be carried into the working places; the cartridges may be prepared at a central and convenient place. This permits a considerable saving in labor and transportation and reduces the wear and tear on the vacuum equipment, and permits the use of larger and more

efficient containers and makes possible the saving of a large amount of oxygen which would otherwise be lost. Fig. 11 shows the method of soaking the cartridges underground.

Strength of L. O. X.

The strength of any explosive is a function of its total energy and also a function of the rate of release of that energy, or in other words, the rate of detonation. The two methods most commonly used for determining the total energy are the ballistic pendulum and the Bichel gage, or closed bomb.

Ballistic Pendulum

Many tests were carried out using the ballistic pendulum in the Bureau of Mines explosive laboratory at Pittsburgh. For a description of this ballistic pendulum, and its method of use, see Bureau of Mines Bulletin 15.[3] First a unit charge is fired of 227 g. of the Bureau of Mines standard 40 per cent. dynamite, the swing of which is 3.4 in. A known quantity of the explosive to be tested is then fired and from a comparison of the swings obtained, the weight of this explosive which would give the same swing as the standard charge is calculated in direct proportion.

This value is known as the unit deflective charge (u. d. c.) of the explosive in question. Owing to the fact that L. O. X. is continually changing in weight, due to the evaporation of the oxygen, it was found better to use a unit deflective volume (u. d. v.) rather than a unit deflective weight. The volume of the unit charge of the bureau's standard 40 per cent. dynamite is 172 c.c. Table 5 gives the average number of cubic centimeters of cartridge of various materials required to give the same pendulum swing as 172 c.c. of the standard dynamite. These results indicate that carbon is the best of the absorbent materials tested, and equivalent to 40 per cent. dynamite.

Table 5.—*Ballistic Pendulum Results with Typical Cartridge Materials*[a]

Material	Volumetric u. d. c. in 1 c. c.
Straight carbon of various brands	(156 to 222)
Wood meal	171
Carbene	173
Carbon, 80 per cent.; ferrosilicon, 20 per cent	167
Carbon, 80 per cent.; aluminum, 20 per cent	169
Carbon, 80 per cent.; infusorial earth, 20 per cent	187
Carbon, 80 per cent.; naphthalene, 20 per cent	205
Standard dynamite	172

[a] The above cartridges were all of densities of from 0.29 to 0.35 and were fired within 5 min. after removal from the oxygen.

[3] Clarence Hall, W. O. Snelling and S. P. Howell: Investigations of Explosives Used in Coal Mines, with a chapter on Natural Gas Used at Pittsburgh by G. A. Burrell, and an introduction by C. E. Monroe. U. S. Bur. of Mines *Bull.* 15, 1911.

Fig. 15 shows the relation between the pendulum swing and the weight of carbon fired, the percentage of oxygen in the cartridge at the time of firing being the same in all tests. As this curve is a straight line, it shows that the swing or strength is strictly proportional to the amount of carbon fired.

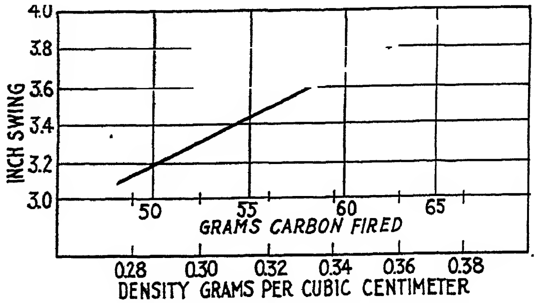


Fig. 15.—Variation in pendulum swing with weight of carbon fired $2\frac{1}{8}$ by 3-in. cartridges of various densities. Percentage of oxygen in the cartridge when fired was the same in all tests.

Fig. 16 shows the results of tests to determine whether the diameter of the cartridge had any effect on strength. This curve shows that the swing and strength are independent of the diameter of the cartridge, at least within the range of size tested.

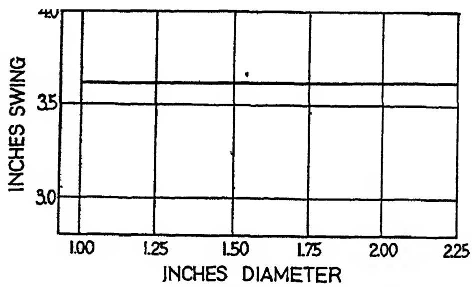


Fig. 16.—Variation in pendulum swing with diameter of cartridge. All cartridges were of same weight and fired with same amount of oxygen present.

Bichel Gage

In the Bichel gage, or closed bomb, various cartridges were exploded. Table 6 gives the results of these determinations. With L. O. X. cartridges, the petcock on the bomb is kept open until the instant of firing when it is closed, thus leaving the bomb filled with oxygen. Since insertion of the cartridge, closing the bomb and getting ready for firing takes 10 min., L. O. X. cartridges cannot be fired in this device at their theoretically best time. The only conclusion to be drawn from the

tests is that the carbon L. O. X. cartridges are superior to the other mixtures, and equivalent to 40 per cent. dynamite, volume for volume.

TABLE 6.—*Volume of Gases and Pressure in Own Volume Generated by the Explosion of Cartridges of Various Composition in a Closed Bomb*
All Cartridges 1¼ by 8 In., Fired at 10 Min.

Material	Volume of Gases, Liters N. T. P.	Pressure in Own Volume, Kg. per Sq. Cm.
Carbon	85.8	95.7
Wood meal	61.9	91.9
Carbene	70.5	76.0
Carbon, 80 per cent.; ferrosilicon, 20 per cent	87.4	78.1
Carbon, 80 per cent.; aluminum, 20 per cent	81.2	79.6
25 per cent. gelatine dynamite	82.2	88.2
40 per cent. gelatine dynamite	89.5	99.1

Rate of Detonation of L. O. X.

The determination of the rates of detonation of L. O. X. cartridges were all made by the lead-plate and T. N. T. cordeau, or Dautriche method.

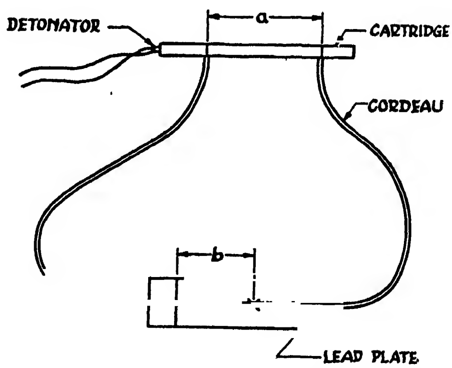


FIG. 17.—DAUTRICHE METHOD FOR DETERMINING RATE OF DETONATION.

In order to determine the rate of detonation by this method, a piece of cordeau, whose rate of detonation is known, is taken and its exact center marked. The cordeau is bent into a loop and the two ends inserted into the cartridge a measured distance apart as shown in Fig. 17. The cordeau is so arranged that the center mark coincides with a line drawn near the end of a 12 by 4 by ¼ in. lead plate. The cartridge is then detonated from one end. As the explosive wave travels through it, the cordeau is in turn detonated first from one end and then from the other.

These explosive waves meet at some point, which after a little experience can be readily determined by the markings on the lead plate. The rate of detonation of the L. O. X. can then be determined from the formula,

$$\frac{\text{Rate of L. O. X.}}{\text{Rate of cordeau}} = \frac{a}{2b}$$

Fig. 18 shows the relation between the rate of detonation and the time of firing, or the influence of the amount of oxygen present upon the rate of detonation.

Table 7 shows the rate of detonation for cartridges of different size and density. These results show that the rate is not appreciably affected by either size or density.

Table 8 shows the rate of detonation of L. O. X. cartridges made of different materials. In this table, all absorbents which will absorb

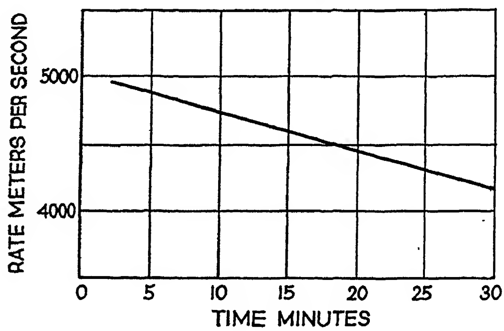


Fig. 18.—Variation in rate of detonation with time of firing; 1½ by 8-in. carbon cartridges, wt. 67 g.

enough oxygen to reach the CO_2 point were detonated at the CO_2 point, and other substances, such as coal dust which will not absorb sufficient oxygen to reach the CO_2 point, were fired when retaining their maximum amount of oxygen. This table again demonstrates that the carbon blacks have the highest rate of detonation.

Table 7.—*Rate of Detonation at the CO_2 Point of Straight-carbon Cartridges*[a]

Diameter, in.	Density 0.26	Density 0.28	Density 0.30
	M. per Sec.	*M. per Sec.*	*M. per Sec.*
1⅛	5650	5520	5640
1¼	5570	5640	5930
1⅜	5610	5600	5730
1½	5950	5580	5550

[a] There is no appreciable variation in rate with density within the limits of the densities that are practical.

There is no variation in rate with diameter, at least in the smaller sizes.

TABLE 8.—*Rates of Detonation of Various Materials*

	M. per Sec.
Carbons	4700 to 5800
Naphthaline	2600
Sawdust	3200
Sulfur	4300
Coal dusts (20 mesh)	2100 to 2900
Carbazol	3500
Carbon (4900), 50 per cent.; sodium bicarbonate, 50 per cent	4200
Carbon (4900), 50 per cent.; sodium chloride, 50 per cent	4700
Carbon (4900), 50 per cent.; potassium nitrate, 50 per cent	4300
Carbon (4900), 50 per cent.; ferrosilicon, 50 per cent	4300
Carbon (4900), 50 per cent.; aluminum, 50 per cent	3800
Carbon (4900), 50 per cent.; naphthalene, 50 per cent	4400
Carbon (4900), 80 per cent.; infusorial earth, 20 per cent	4400
Naphthalene 80 per cent.; infusorial earth, 20 per cent	2200
Cellulose (paper)	1600
Carbon (5600), 90 per cent.; mineral oil, 10 per cent	5600
Carbenes	4700 to 5000
Mixtures of carbene with various other materials give no results of importance.	

Continuity of Detonation

Tests were made with a string of cartridges, 1¼ by 12 in., in the open air laid end to end in contact. The line was 40 ft. long and was detonated by a cap in the end cartridge. All the cartridges detonated uniformly.

Detonation by Influence

Cartridges 1¼ by 12 in. were laid on the ground end to end with an air gap between. It was found that straight carbon cartridges would detonate in this way across an air gap of from 5 to 7 in. By the addition of suitable hydrocarbons, or metallic powders, the gap across which they will detonate can be increased to 20 in. or more. Tests made on 1¼ by 12-in. cartridges in a drill hole, placing one cartridge in the bottom of the hole and detonating a cartridge at the collar, demonstrated that the bottom cartridge can be detonated across a gap of 3½ ft. or more. Evidently no difficulty in mining will occur because of failure of the cartridges to detonate should they become slightly separated in the hole.

MISCELLANEOUS TESTS

It was found impossible to detonate any of the carbon L. O. X. cartridges in the open air by flame only, that is, by a fuse inserted in the cartridge. Caps were necessary to secure detonation. However, when confined in a drill hole, the cartridges can be detonated by a fuse without cap, and this is the standard practice in the iron mines of Lorraine.

Table 9 gives the effect on the rate of detonation by various detonators. The conclusion from this is that the detonators are practically all alike, so far as the rate of detonation of the cartridges is concerned, and there is no reason why No. 6 cap should not be just as effective as No. 8.

Table 10 shows the result of an interesting test to determine how small the amount of oxygen in the cartridge could be, and still secure detonation. In these tests, detonation was secured when 3.3 per cent. of the oxygen required for combustion to CO_2 was present. As the oxygen evaporates, there is a tendency to leave an oxygen saturated core in the center of the cartridge, and detonation can be obtained as long as this core is present, provided the cap is in contact with it. A 4-in. cartridge detonated at the end of a 4-hr. period in the open air.

TABLE 9.—*Rate of Detonation of L. O. X. with Various Detonators*

No.	Detonator	Rate M. per Sec.	No.	Detonator	Rate M. per Sec.
5	Electric	4900	8	Lead azide	5090
6	Electric	4990	10	Mercuric fulminate	4990
7	Electric	4880	8	Mercuric fulminate	4990
6	Tetryl electric	4950	6	TNT picric acid	4856
8	Tetryl electric	4990		Mercuric azide	4950

TABLE 10.—*Detonation with Small Amounts of Oxygen Present*

1⅜ by 12-in. Cartridge, Dry Weight 85 g. Straight Carbon

Time out of Liquid before Firing, Min.	Weight of Cartridge, G.	Result
10	211	Good detonation, some black smoke.
15	204	Good detonation, some black smoke.
20	178	Good detonation, some black smoke.
25	164	Good detonation, slightly more smoke.
30	146	Good detonation, slightly more smoke.
35	132	Good detonation, considerable unburned carbon.
40	109	Good detonation, much unburned carbon.
45	105	Report weaker, much unburned carbon.
50	103	Report weaker, some unconsumed paper.
55	99	Report weaker, some unconsumed paper.
60	92	Report weaker, some unconsumed paper.
65	90	Cap only.

Tests were made to determine the sensitiveness of L. O. X. to impact. These tests were made on the large impact testing machine at the Bureau of Mines explosive laboratory. In this machine, the explosive is placed between steel discs and a weight of 200 kilograms is dropped on it from varying heights. The sample of L. O. X. used was ¼ in. thick, and 3¾ in. in diameter, weighing approximately 35 g. Table 11 shows the results obtained.

TABLE 11.—*Results of Impact Tests of L. O. X. Cartridges*
Large Impact Machine, 200 Kg. Weight
L. O. X. in Layer ¼ In. Thick, 3.75 In. Diameter, between Two Steel Discs. Weight Taken Approx. 35 g.

Material	Maximum Drop without Explosion, Cm.	Minimum Drop with Explosion, Cm.
Carbon	15	20
Carbene	10	20
Carbon, 80 per cent.; ferrosilicon, 20 per cent	..	2.5
Carbon, 80 per cent.; aluminum, 20 per cent	20	30

In addition to these impact tests, many attempts were made to detonate L. O. X. by throwing cartridges against rock faces in stopes at distances of from 60 to 80 ft. In every test the cartridges failed to detonate.

A test was made allowing a lighted acetylene miner's lamp to drop into a container full of prepared cartridges. This failed to detonate them. When a container of soaked cartridges is set on fire, however, the flame is very large and intense.

It is possible to detonate a container full of prepared cartridges by shooting through the container with a rifle bullet.

Friction Tests

Tests were made in the friction-testing machine at the Bureau of Mines explosive laboratory; all tests either with the fibre shoe or the steel shoe failed to detonate L. O. X.

COMPARISON OF L. O. X. WITH OTHER EXPLOSIVES

Table 12, furnished by the Bureau of Mines, shows the properties of some of the explosives in common use.

TABLE 12.—*Properties of Some Common Explosives*[a]

Explosive	Maximum Height of "No Explosion" in Large Impact Machine, Cm.				Passed Friction Test with Fibre Shoe	Rate of Detonation, M. per Sec.		U. D. C., G.	
	Confined		Unconfined						
	From	To	From	To		From	To	From	To
25 per cent. gelatine	5	5	5	5	Yes	1633	1785	270	297
40 per cent. gelatine	2.5	5	2.5	2.5	Yes	4943	4972	231	280
40 per cent. straight[b]	5	10	5	15	Yes	4060	5288	231	271
40 per cent. ammonia	2.5	60	5	40	Yes	3384	4442	228	253
60 per cent. gelatine	2.5	2.5	10	10	Yes	5311	6316	206	227
60 per cent. straight					Yes	5716	6246	209	211

[a] Courtesy of U. S. Bureau of Mines.
[b] Straight dynamite primer used with the 60 per cent. gelatine.

TABLE 13.—*Composition and Properties of Bureau of Mines Standard Dynamite,* 40 *Per Cent.*

	Per Cent.
Composition:	
Nitroglycerine	40
Sodium nitrate	44
Wood pulp	15
Calcium carbonate	1
	100

Weight per unit charge, 227 g. (U. d. c.)
Volume per unit charge, 172 c.c. (U. d. v.)
Density 1.32 g. per c.c.
Average pendulum swing:

	In.
Per charge	3.40
Per gram	0.0150
Per c.c	0.0198
Per pound	6.80

In testing the strength of L. O. X. it was decided to use as a basis of comparison the Bureau of Mines standard 40 per cent. dynamite

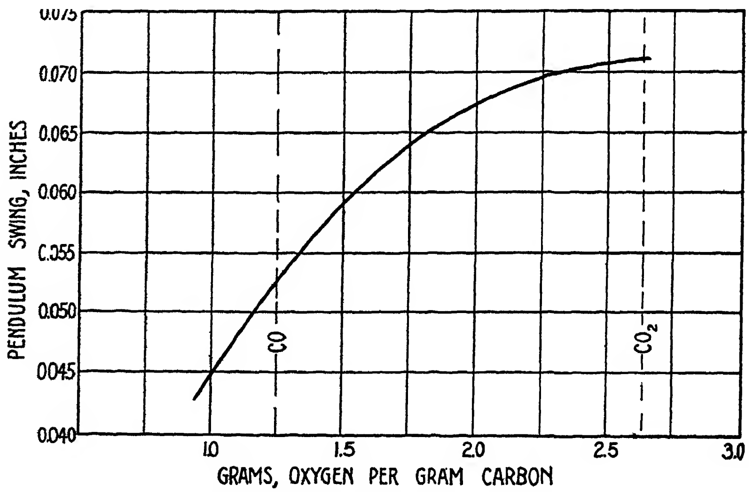


FIG. 19.—BALLISTIC PENDULUM SWING DUE TO 1 G. CARBON BLACK COMBINED WITH VARIOUS WEIGHTS OF OXYGEN.

which has the composition and properties shown in Table 13. Our first tests were to determine the effect on the strength of the cartridge of variation in diameter, density of packing and amount of oxygen present.

Figs. 15 and 16 show that the strength is not influenced by the size of cartridge or its density of packing, but depends only upon the amount of carbon detonated and the amount of oxygen present at the time of detonation. These facts being known, our data were reassembled on the basis of pendulum swing per unit of carbon and the amount of oxygen associated with it.

Fig. 19 gives the pendulum swing due to one gram of carbon associated with various amounts of oxygen and is applicable to cartridges of all diameters and weights. Fig. 20 shows swing per pound of carbon for use with large cartridges.

Figs. 3 and 4, and similar curves at different densities were then replotted giving Figs. 21 to 25, which show the grams of oxygen present per gram of carbon for various diameters, densities and lengths of time. A density of 0.3 was chosen as best for large cartridges. These curves

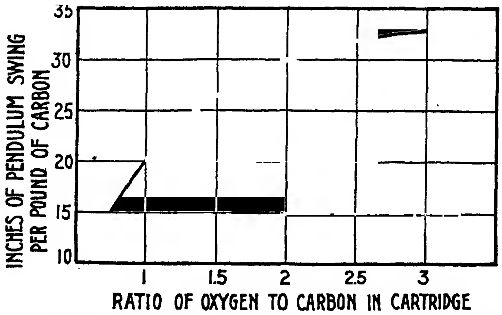


Fig. 20.—Curve shows pendulum swing due to 1 lb. of carbon fired with various amounts of oxygen.

make it possible to predetermine the performance of a cartridge if we know its weight, its size and the time of firing. Many of these calculated swings are, of course, beyond the range of the ballistic pendulum. They afford, however, a satisfactory basis for numerical comparison. For example, let us assume that a $1\frac{1}{4}$ by 12-in. cartridge weighing

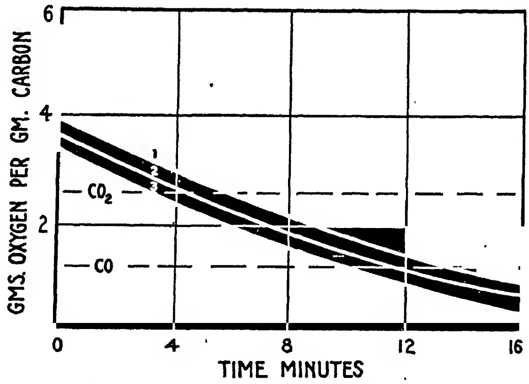


Fig. 21.—Cartridges, $1\frac{1}{8}$ by 10 in.; No. 1 density, 26; No. 2 density, 28; No. 3 density, 30.

77 g. is to be fired in a drill hole at a time equivalent to 10 min. in the open air. The density of the cartridge is 0.32. From Fig. 22, the grams of oxygen present per gram of carbon is 1.64; from Fig. 19, the pendulum swing per gram of carbon is 0.061. The pendulum swing for the cartridge is then 0.061 multiplied by 77, or 4.69 in. The pendulum

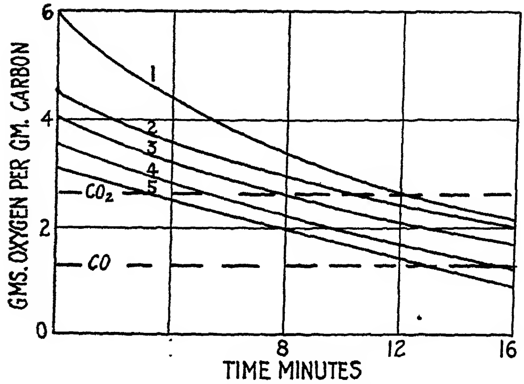


Fig. 22.—Cartridges, 1¼ by 12 in.; No. 1 density, 0.17; No. 2 density 0.21; No. 3 density, 0.24; No. 4 density, 0.28; No. 5 density, 0.32.

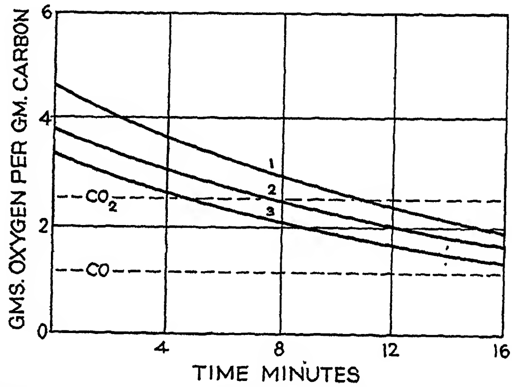


Fig. 23.—Cartridges, 1⅜ by 10 in.; No. 1 density, 0.21; No. 2 density, 0.27; No. 3. density, 0.30.

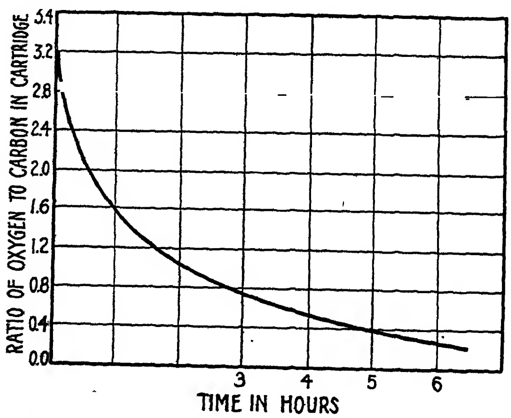


Fig. 24.—Cartridges, 7 by 20 in; carbon in canvas bag; density, 0.29.

swing due to an equal volume of standard 40 per cent. dynamite is 4.77 in. from Table 13.

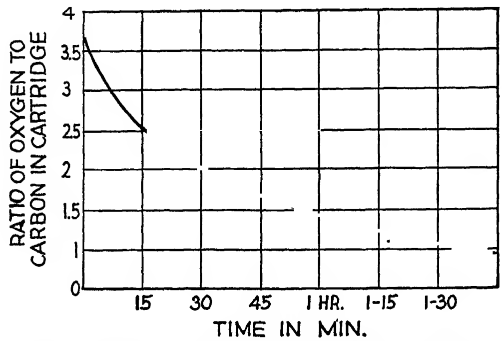


FIG. 25.—CARTRIDGE, 4½ BY 18 IN. × 3 LB; CARBON BLACK IN COTTON BAGS.

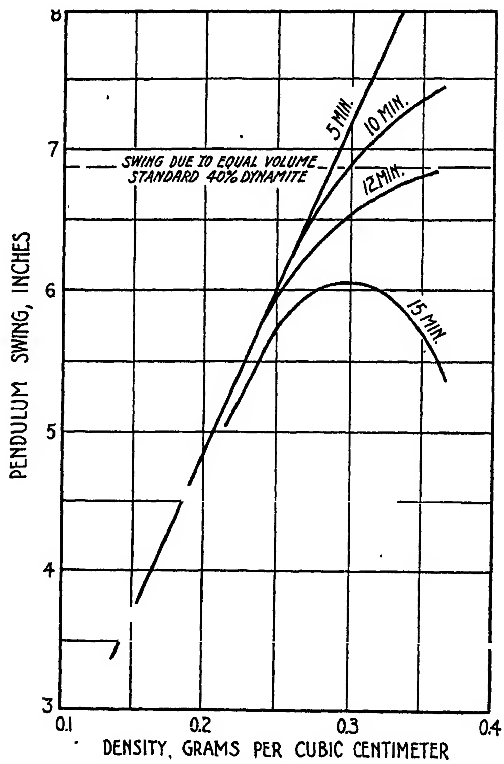


FIG. 26.—VARIATION IN PENDULUM SWING WITH DENSITY AND TIME 1½ BY 12-IN. CARTRIDGES.

The above results are based on our standard No. 2 carbon (29*a* of Table 2). Many additional tests of a similar nature were made with

other carbons, and with numerous mixtures of carbon with metallic powders, hydrocarbons, and other substances, but nothing was found that was superior to No. 2 carbon.

Figs. 26 to 28 show the pendulum swing at various densities, and at different times of firing. These curves show the relation between the performance of equivalent volumes of L. O. X. and dynamite, and demonstrate that for the smaller cartridges with densities of approximately

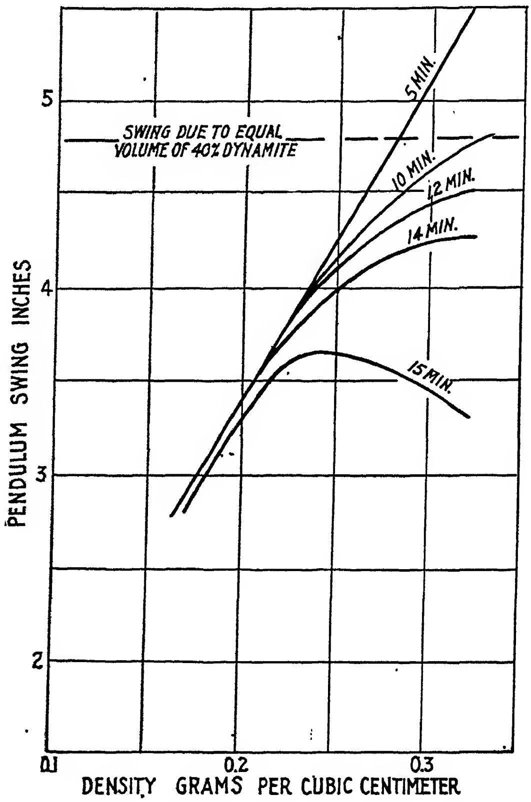


FIG. 27.—VARIATION IN PENDULUM SWING WITH DENSITY AND TIME; 1¼ BY 12-IN. CARTRIDGES.

0.3, firing at 10 to 12 min., L. O. X. is equivalent to 40 per cent. dynamite volume for volume. For the larger cartridges, the same holds for firing at 30 to 45 min. These firing periods are, of course, based upon the loss of oxygen which takes place in the open air, and are extended considerably when the cartridges are in the drill hole.

Tests were made to determine the additional length of life, or reduction in loss of oxygen when cartridges were in the drill holes as compared to the losses in the open air. Fig. 29 shows some tests on 1⅜ by 12-in.

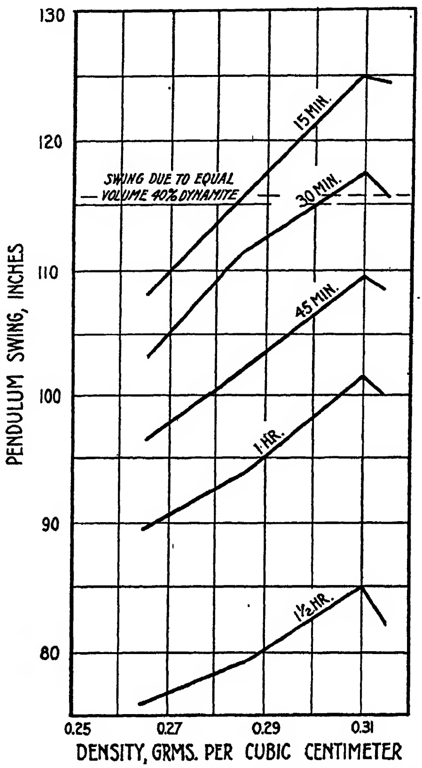


FIG. 28.—VARIATION IN PENDULUM SWING WITH DENSITY AND TIME; 5 BY 18-IN CARBON CARTRIDGES.

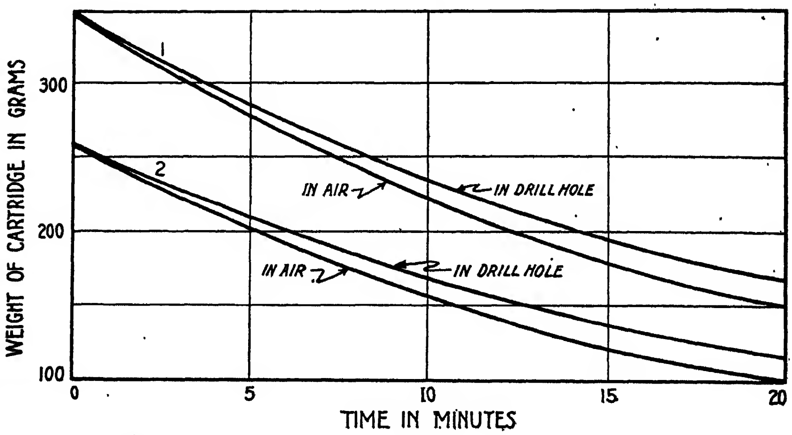


FIG. 29.—COMPARISON OF TYPICAL TIME-WEIGHT EVAPORATION CURVES IN AIR AND IN DRILL HOLES. No. 1 (1⅜ BY 12-IN.) CARBON CARTRIDGES, DRY WEIGHT 85 G.; No. 2 (1¼ BY 10-IN.) CARBON CARTRIDGES, DRY WEIGHT 58 G. BOTH IN PAPER WRAPPER. HOLES 6 FT. DEEP IN GNEISS; 5 CARTRIDGES PER HOLE; 9 IN. CLAY TAMPING. AVERAGE INCREASE DUE TO DRILL HOLE, 15 PER CENT.

and 1¼ by 10-in. cartridges in air and in drill holes, and indicates that the life is extended from 15 to 20 per cent.

As the foregoing data demonstrate that L. O. X. and 40 per cent. dynamite are equivalent volume for volume, it is now possible to compute the weight of carbon and the weight of oxygen needed to replace dynamite, provided that the L. O. X. cartridges can be fired at such times as to maintain this relation. We will assume a cartridge density of 0.3 as this has been found in practical work to give the most satisfactory results. One pound of carbon black packed to 0.3 density occupies a volume of 92 cu. in. One pound of 40 per cent. dynamite occupies a volume of 21 cu. in. Therefore, the pounds of carbon needed per pound of dynamite is 21 divided by 92, or 0.23. In other words, 0.23 lb. of carbon at 0.3 density occupies the same volume as a pound of 40 per cent. dynamite, and the volume of the L. O. X. cartridge is determined, of course, by the volume of the carbon. Our practical experience underground has demonstrated that where the work in stopes and drifts is so located that the soaked cartridges are transported for a relatively short distance, we have to deliver from the oxygen column 6 lb. of oxygen per pound of carbon, or 1.38 lb. of oxygen per pound of dynamite. Where the cartridges have to be transported underground a mile more or less, we require 7 lb. of oxygen per pound of carbon, or 1.6 lb. of oxygen per pound of dynamite.

Practical Applications of L. O. X.

An oxygen plant with a capacity of 75 liters of liquid per hour was operated experimentally at the Witherbee-Sherman mine at Mineville, N. Y., from November, 1923, to October, 1924. The plant was located underground approximately 1000 ft. below the surface and operated satisfactorily throughout the test. Figs. 30 and 31 show views of this plant. The ore in this mine is magnetic oxide of iron very definitely crystallized; it occurs in large masses. The mine is worked in large breast stopes, pillars being left to support the roof. The country rock is a hard gneiss with bands of magnetic oxide, and both the ore and the rock are extremely abrasive so that the mine practice is to use ¼-in. changes of gage for each 2-ft. change in length of drill steel as shown below:

	Ft. In.		Ft. In.
Stopes:		Drifts:	
First steel	2 by 2	First steel	2 by 2¼
Second steel	4 by 1¾	Second steel	4 by 2
Third steel	6 by 1½	Third steel	6 by 1¾
Fourth steel	8 by 1¼	Fourth steel	8 by 1½

Figs. 32 and 33 show typical arrangements of holes in drifts and stopes. The nominal depth of all holes both in drifts and stopes is 8 ft., but in actual practice they are rarely over 7½ ft.

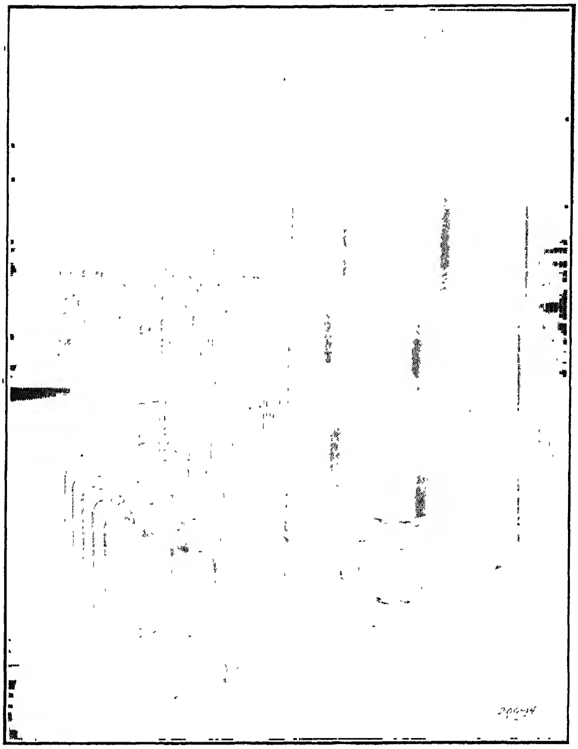


FIG. 30.—SEVENTY-FIVE-LITER OXYGEN PLANT SHOWING SODA TOWERS, DESICCATORS AND MIXING TANK.

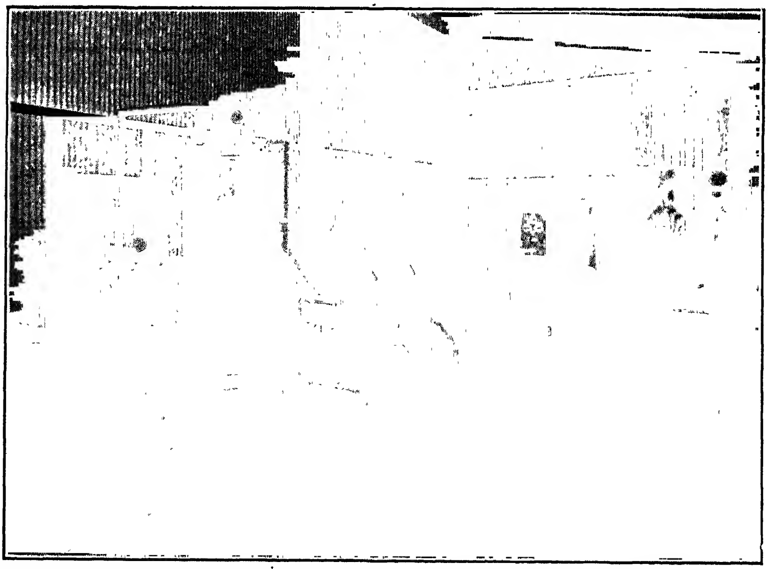
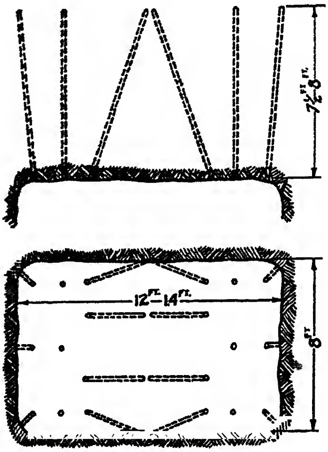


FIG. 31.—SEVENTY-FIVE-LITER OXYGEN PLANT, SHOWING OXYGEN COLUMN, COMPRES-

Practically all of the drilling in stopes is done with Ingersoll-Rand CR-430 jackhammers; the drilling in the drifts is done with Ingersoll-



DRIFT ROUND

FIG. 32.—TYPICAL DRIFT ROUND AT WITHERBEE-SHERMAN CO. MINES.

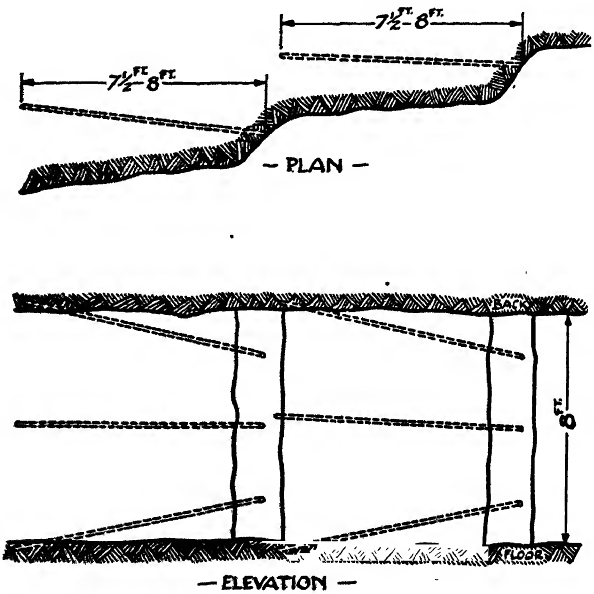


FIG. 33.—TYPICAL STOPE ROUND WITHERBEE-SHERMAN CO. MINES.

and 248 Leyner drills. On account of loss in gage, the maximum O. X. cartridge which could be used in the 8 ft. stope holes is 1⅛ in. d in the 8-ft. drift rounds 1⅜ in. and sometimes 1¼ in.

Loading and Firing

Perhaps the best way to describe the methods of loading and firing that were developed, will be to give the difficulties encountered and the means by which they were overcome. The handling of oxygen containers and soaking vessels in the drifts and stopes was eliminated by the "dry carrier." All cartridges were soaked at the oxygen plant, and then were distributed in dry carriers to the working places with very satisfactory results.

Electric firing was tried to some extent, and was satisfactory, but was found too cumbersome for use in the stopes. Also it was not considered advisable because of the large amount of electricity used in the mine and the heavy stray currents present in practically all pipe lines and rails.

In firing with cap and fuse, we first used the method employed at Pachuca of placing the cap in a small clay or wood plug in the bottom of the hole. Although this gave very satisfactory results in the comparatively shallow holes used at Pachuca, the 8-ft. holes at Mineville required

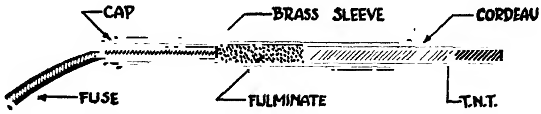


FIG. 34.—SECTION THROUGH CAP PROTECTOR AND BOOSTER.

fuses of from 9 to 10 ft. long, and too much time was consumed while the fuse was burning. We are also of the opinion that this is not the most effective position for the cap.

Attempts were then made to place the caps directly in the cartridge as is done with dynamite. This involves several difficulties. The fuses must be cut to the proper length to give the correct order of firing after the holes are loaded. Much valuable time is thus lost, and an operation which should be done carefully and deliberately is hurriedly and improperly performed. If the fuses are cut beforehand, miners will often become confused with the various lengths and the holes are not exploded in the order desired. If the caps are inserted in the cartridges while loading, there is further loss of valuable time, while if they are placed in the cartridges beforehand and soaked with them, misfires will result, as fulminate of mercury caps frequently fail to detonate properly when chilled to the temperature of liquid oxygen.

These difficulties were overcome by the use of the device shown in Fig. 34. This consists of a short brass sleeve to cover the cap and connect to it a 1 or 2-in. length of scrap cordeau. These can be placed at any point in the hole, alongside the cartridge. If properly crimped, the cartridges slide by them readily, and the results obtained are very satisfactory. The brass sleeve protects the cap from being rubbed against

the side of the hole. The small cordeau booster gives more effective detonation of cartridges low in oxygen. These boosters are placed in the hole, and the fuses cut ready to spit before the holes are loaded, which eliminates the need of inserting the cap in a cartridge.

Stemming

The first stemming used was moist sand in the usual paper tamping-bags. This was not satisfactory, due to its nonplastic character. The cartridges did not fit the hole tightly and the cartridges were thrown out of some of the holes by the explosion in those adjacent to them.

A sticky plastic clay wrapped in waxed paper was then tried, and gave very satisfactory results. No difficulty was experienced with this stemming blowing out, provided it was not tamped too tightly, and a cartridge was used which held the oxygen properly. When loading, it is advisable to remove the basket from the carrier, and allow the cartridges to drain a few seconds before placing them in the holes. If the holes are wet, the cartridges lose their oxygen more rapidly, and they must be fired sooner; also the stemming is more liable to blow out. These are the only important difficulties with wet holes.

Fireproof Fuse

"Clover Brand" fuse manufactured by the Ensign Bickford Co. of Simsbury, Conn., was first used. This fuse, although satisfactory under most conditions, has the disadvantage that it is not fireproof. Sometimes when holes are located in a depression or near the floor, the air about the collar of the hole becomes so enriched with oxygen that the fuse when lit takes fire and burns as a whole. This difficulty was overcome by the development of a special L. O. X. fuse which is fireproof under all conditions. This fuse can be obtained at a price very little higher than the ordinary Clover Brand. The rate of burning of this fuse is about 42 sec. per foot. The rate is not appreciably affected by the oxygen in the drill hole. We have had no trouble from fuses side-spitting.

After the fuses have been prepared and placed in the holes, the actual loading, with 6 to 8-ft. holes, 5 cartridges per hole, can be done at the rate of a hole per man per minute. Two men have loaded 120 cartridges into 22 holes with 6-ft. fuses in 13½ min. from the time the first cartridge was removed from the carrier until the last report.

Certain precautions should be observed when loading. Lights must be kept away from the collars of the holes and, as far as possible, out of line with the holes. Cap lamps should not be worn. This is necessary, because cartridges often break in the hole, and the oxygen escaping through the stemming may carry inflammable carbon with it. Also, should the stemming blow out, the cartridge usually comes out with it and would, of course, catch fire should it fall on an open light.

The length of fuse outside the hole should be ample, and the ends should be fastened in place with clay, well clear of the collars and of all cracks. If the fuses are not fastened in place, there is danger that when a fuse is lit, it may curl into the collar of a hole and set fire to it. Spitting should be carefully done so that lamps are not brought into contact with any broken cartridges of blown-out carbon on the floor or face. Leather gloves, although not absolutely necessary, are advisable when handling the soaked cartridges. Figs. 12 and 13 show the preparation and Fig. 35 the loading of the cartridges.

Fig. 35.—Miner loading L. O. X., shows cap and fuse in position in hole and basket of cartridges in dry carrier.

Comparison with Dynamite

The standard dynamite used at Mineville is 25 per cent. gelatine. By referring to Table 12, it will be noted that in the ballistic pendulum this has 89 per cent. of the strength of the standard 40 per cent. dynamite. The rate of detonation is, of course, much lower.

Results with 8-ft. Holes in Drifts and Stopes

In drifting, it is the mine practice to use in the cut holes about 22 sticks of dynamite 1⅛ by 8 in., a volume of 172 cu. in. In other words, they fill the hole to within a few inches of the collar.

With L. O. X. it was not possible to get into these 8-ft. holes a volume anywhere near equal to the volume of dynamite used. This is due to the fact that the dynamite is plastic and can be squeezed out or upset in

the tapered holes, while L. O. X. cartridges are rigid and cannot be upset. The maximum amount of L. O. X. that could be used in these holes was six 1⅜ by 12-in. cartridges, a volume of 108 cu. in., and the breaks obtained were very poor. This is one of the limitations to the practical use of L. O. X. and we have not yet succeeded in perfecting a method which will permit upsetting an L. O. X. cartridge in a drill hole. As already shown, the standard L. O. X. cartridges can be loaded very quickly, but any method of upsetting them will have a disadvantage in that the extra time required for this work entails an additional loss of oxygen. For example, cartridges were enclosed in cotton bags, in which they could be quite successfully upset to a degree which increased the volume of L. O. X. per foot of drill hole about 20 per cent., but the practical results obtained showed no advantage due to the extra time required in loading.

Another disadvantage of L. O. X. in a tapered drill hole is the fact that the cartridges do not entirely fill the space as the plastic dynamite does. In other words, an air space surrounds the cartridges. The extent of this disadvantage has not been definitely established, but the authors are of the opinion that it is of considerable importance. An indication of its importance was discovered in the use of the ballistic pendulum, where a given weight and volume of L. O. X. tightly fitting the bore of the cannon gave evidence of a much greater disruptive effect on the cannon than did the same weight and volume of L. O. X. which did not fit the bore tightly.

In the 8-ft. stope holes, the mine practice is to use an average of 9¼ sticks of 25 per cent. dynamite, 1⅛ by 8-in., or a volume of 74 cu. in., which gave about 90 to 95 per cent. break. The maximum volume of L. O. X. that could be used in these holes was six 1⅛ by 12-in. cartridges, or about 72 cu. in., the firing time being 12 min., and the average break obtained about 65 per cent. The reason for this will now be shown. These practical results are found to be consistent with the data of tests with the ballistic pendulum. First for the 8-ft. stope holes, 9¼ sticks of dynamite or 4.61 lb., give a pendulum swing (Table 13) of 4.61 in. $\times$ 6.8 = 31.3 in. This multiplied by 0.89 to allow for the difference between 40 and 25 per cent. dynamite gives 27.9 in. A 1⅛ by 10-in. cartridge weighing 50 g., density 0.26, fired at 12 min. (Fig. 21) will have in it at the time of firing 1.50 g. of oxygen per gram of carbon, and will give a swing of 0.058 in. (Fig. 19) per gram, or 2.9 in. for the cartridge. The swing for the six sticks of L. O. X. used is therefore 2.9 in. $\times$ 6 = 17.4 in. as compared to 27.9 in. for the dynamite. This explains the practical results obtained.

In the 8-ft. holes in the drift, the pendulum swing for 22 sticks, or 11 lb., of dynamite is 75 in., or for 25 per cent. dynamite 75 in. $\times$ .89 = 66 in. A 1⅜ by 10-in. cartridge containing 87 g. of carbon, density 0.3, fired at 12 min. will have in it 1.65 g. of oxygen per gram of carbon (Fig. 23) and give a swing of 0.0615 in. per gram, or 5.35 in. per cartridge

(Fig. 19), or 32.10 in. for the six cartridges used as compared to 66 in. for the dynamite.

It may be asked that if the experimental data predicted these results, why were the practical tests made. The answer lies in the fact that our experimental work did not precede all of the practical tests but was carried on simultaneously with them. Furthermore, all of these comparisons are made on the basis of the ballistic pendulum results alone, ignoring the question of rate of detonation. Since the rate of detonation of the L. O. X. greatly exceeds that of 25 per cent. dynamite, it was felt that the only way to get conclusive results was by actual rock-breaking tests. It is the opinion of the authors that the advantage of the rate of detonation of the L. O. X. is offset by the fact that it does not fill the drill hole.

Results with 6-ft. Holes in Stopes

The above results both practically and theoretically demonstrate that 1⅛-in. cartridges fired at 12 min. in 8-ft. holes are inadequate, due to the impossibility of filling the hole with L. O. X. It was, therefore, decided to make a series of practical tests with 6-ft. holes in the stopes, using the standard mine practice of drilling with a jackhammer and standard steels as follows: 2 ft. by 2 in., 4 ft. by 1¾ in., 6 ft. by 1½ in.

Due again to the abrasive nature of the rock, it was not found possible to use 1⅜-in. cartridges in these holes. In many tests attempts to use 1⅜-in. cartridges developed that the hole was too small, the cartridges would not bottom and the hole was lost. Therefore, 1¼-in. cartridges had to be adopted.

This trial extended over several weeks, and involved the shooting of approximately 1000 holes, the rounds being anywhere from 6 to 24 holes at a time. An average of five 1¼ by 12-in. cartridges were used in these holes and the average break obtained was about 90 per cent., that is practical results were obtained. Tables 14 and 15 are based on work in two stopes where the conditions were practically identical and give the relative cost of the standard mine practice, using 8-ft. holes as compared with our results using 6-ft. holes. It will be noted that the cost of explosives per ton with L. O. X. is considerably more than with dynamite, under mining conditions existing at this property.

TABLE 14.—*Summary of Operations in 26B Stope, Harmony Mine*

L .O .X. 1¼ In. by 12 In. Carbon Cartridges, 6-ft. Holes, Fair Stoping Conditions in Lean Ore

Machine shifts	20	Explosives:—	
Holes fired	186	Oxygen used 745 lb. @ $0.0229	$17.06
Total feet of hole	1,083.5	Cartridge used 134.5 lb. @ $0.024	32.28
Feet of hole broken	951.9	Boosters used 186 @ $30.00 M	5.58
Per cent. break	88		
Tons broken	618		$54.92
Tons per machine-shift	30.9	Cost per ton for explosives	$ 0.0888

TABLE 15.—*Summary of Operations in 27B Stope, Harmony Mine*

25 Per Cent. Gelatine Dynamite 1⅛ by 8 In. Cartridges, 8 Ft. Holes, Fair Stoping Conditions in Lean Ore

Machine shifts	17	Tons broken	840
Holes fired	75	Tons per machine-shift	49.4
Total feet of hole	582	Explosives:—	
Feet of hole broken	553	25 per cent. gelatin 410 lb. @	
Per cent. break	95	$0.1375 lb	$56.38
		Cost per ton for explosives	$ 0.0671

Results with Larger Cartridges

It was next decided to try the effect of 8-ft. holes with larger cartridges. No. 248 Leyner machines were placed in one of the stopes and 8-ft. holes were drilled with diameters large enough to permit the use of 1⅜, 1½, and 1¾-in. cartridges. This test was carried on for about 2 mo. The results, however, were not favorable. With cartridges of this size the cost of the explosive per ton of rock broken was higher than with dynamite.

Discussion of Results

The authors attribute the failure at Mineville to the fact that the ground is so hard and abrasive, resulting in so much loss in gage in drill steel as to cause the holes to be considerably tapered. Due to the non-plastic nature of L. O. X., it is not possible to fill the tapered hole to the equivalent volume of dynamite. This results in two losses: First, the direct loss of mass of explosive above mentioned; and, second, the loss of effectiveness due to the fact that the explosive does not fill the hole, which causes a considerable air cushion between the explosive and the walls of the hole. The extent of this latter effect is difficult to determine, but in the opinion of the authors it is considerable, which is borne out by the fact that in large cylindrical holes in bench mining and quarry work where the L. O. X. fits the hole rather closely, the effectiveness is excellent.

Actual rock-breaking tests were also conducted with cartridges of various carbons, hydrocarbons, and in fact almost every conceivable oxidizable substance, but nothing was found superior to our standard black. This work was a practical confirmation of our laboratory work.

Conclusions

In the light of our work at Mineville, the following conclusions may be drawn regarding the use of L. O. X. underground:

1. Where the ground is heavy and abrasive, causing the drill holes to be considerably tapered, L. O. X. costs more than dynamite.

2. Where the method of mining involves the drilling of comparatively shallow holes without a large loss of gage, as in Lorraine, L. O. X. may be commercially profitable, especially where the price of dynamite is high.

3. The use of L. O. X. does away with the necessity of maintaining a stock of explosives at the mine, and renders the powder supply independent of interruptions in transportation.

4. With respect to safety, it has been shown there is no danger of the premature detonation of L. O. X. There is, however, danger of premature ignition resulting in violent long-flame conflagration, which would be dangerous in confined places underground. In loading holes, there is some danger of the liberated gas blowing out the stemming and, perhaps, ignition of carbon-laden gas by miners' lamps. The danger of drilling into missed holes is eliminated, and on this score it is safer than dynamite.

With respect to noxious gases, it is undoubtedly superior to dynamite, although there is no doubt that some CO is produced if the holes are fired late. In all of our work, only one man was affected by CO, which resulted from firing several rounds of shots in succession and immediately returning to a dead end stope. It is the conclusion of the authors that much less CO is formed with a deficiency of oxygen than we had first assumed, probably because the core oxidizes to CO_2 when the cartridge detonates and blows the excess carbon out as unburned carbon, which is always present with late shots. Furthermore, there is probably some oxidization of the CO to CO_2 in the atmosphere at the time of detonation.

In heavy sulfide ores, there will, no doubt, be considerable production of SO_2 with the use of L. O. X.

5. The authors would not recommend the use of L. O. X. in coal mines due to the long hot flame of explosion.

6. When firing must be done simultaneously in a large number of adjacent stopes, it is much more difficult to synchronize this work with L. O. X. than with dynamite.

7. More holes are lost with L. O. X. than with dynamite, due to the fact that the fuse is not held as firmly as with dynamite, and to the fact that since the cartridges fit the hole loosely, they are sometimes jarred out by the detonation of adjacent holes. If a cartridge jams in loading, due to a crooked hole, there is no time to remedy this, and there is usually not time to reload a hole that blows out the stemming during the loading period.

The Use of L. O. X. in Quarries and Open-pit Mines

During the operations at Mineville, from time to time we shipped oxygen to various quarries and obtained such highly satisfactory results that the liquid-oxygen plant was moved to Lebanon, Pa., from which a good many quarries and open-pit operations could be reached. We were also able to make tests in a wide variety of rocks, as Table 16 shows. Our operations were mostly in 6-in. holes, there being a few 2, 4 and one

TABLE 16.—*Data on Blasts Made in Open-pit Mines and Quarries*

Blast No.	Material Blasted	No. of Holes	Distance Apart of Holes, Ft.	Distance Back from Face, Ft.	Time Fired, Min.	Depth of Holes, Ft.	Diameter of Holes, In.	Dynamite Replaced, Pounds	Oxygen Used, Pounds	Cartridge Used, Pounds, (Dry Weight)
1	Limestone	4	12	15	30	28	6	500	360	48
2	Limestone	12	12	12	95	27	6	1,020	1,014	195
3	Limestone	18	12	17	28	23	6	1,980	1,394	276
4	Limestone	18	12	17	30	23	6	2,000	1,224	296
5	Limestone	12	12	12	90	41	6	2,520	1,428	327
6	Limestone	18	12	17	30	23	6	1,620	1,632	353
7	Iron ore	16	10	10	32	42	6	1,771	1,500	352
8	Limestone	18	12	17	50	23	6	1,620	1,440	330
9	Limestone	18	12	17	40	23	6	1,620	1,428	330
10	Limestone	10	10	10	20	18	6	750	714	146
11	Limestone	18	12	17	25	23	6	1,620	1,428	330
12	Limestone	25	12	17	30	23	6	2,250	2,040	543
13	Trap	1		30	30	75	8	1,500	1,410	248
14	Limestone	28	Irregular		30	8–24	6	1,500	1,510	275
15	Limestone	10	10	12	25	19	6	500	740	148
16	Sandstone	30	Irregular		35	22	4	2,300	1,900	335
17	Limestone	40	Irregular		30	7–18	6	1,600	1,596	297
18	Shale	7	8	12	10	6–18	2	70	105	15
19	Shale	1		12	15	26	6	50	140	21
20	Limestone	39	10	10	40	30	6	2,730	2,569	588
21	Limestone	6	12	20	40	40	5½	1,600	970	184
22	Limestone	13	15	27	50	30	6	1,700	1,060	227
23	Limestone	10	12	12	40	40	6	1,000	1,050	276
24	Limestone	6	12	17	32	32	6	855	1,153	239
25	Limestone	5	12	15	50	42	6	750	700	155
26	Limestone	16	12	15	25	33	6	2,400	2,100	486
27	Limestone	2	20	36	45	106	6	4,350	1,395	372
28	Limestone	8	12	15	45	27	6	900	750	162
29	Shale	7	20	20	20	33	6	700	1,040	198
30	Limestone	3	13	25	35	65	6	2,160	1,410	324
31	Shale	6	20	20	30	30	6	600	1,000	162
32	Limestone	22	12	15	50	25	6	2,300	1,600	402
33	Limestone and shale	8	20	20	50	30	6	775	900	186
34	Limestone	7	12	15	15	37	6	700	750	162
35	Limestone	10	12	15	45	25	6	1,000	1,145	249
36	Limestone	9	15	20	55	45	6	2,500	1,900	420
37	Limestone	5	28	15	95	70	6	3,450	2,815	645
38	Limestone	11	12	15	35	24	6	1,000	1,145	249
39	Limestone	7	12	15	25	24	6	735	720	126
40	Limestone	4	12	15	30	24	6	450	350	75
41	Trap	2	20	12	55	53	6	840	750	162
42	Limestone	39	12	12	55	16	6	1,950	3,000	585
43	Sandstone	5	13	22	55	60	7	3,300	2,637	486
44	Limestone	7	12	15	25	24	6	700	720	147
45	Limestone	10	Irregular		35	24	6	1,000	1,080	240
46	Limestone	8	Irregular		60	52	6	2,200	2,000	444
47	Limestone	15	12	15	60	24	6	1,500	1,475	315
48	Limestone	35	12	12	75	16	6	1,750	1,900	420
49	Limestone	10	12	15	35	24	6	1,000	1,100	240
	Totals							73,686	64,393	13,791

8-in. hole, the depths of the holes varying from 6 ft. to 106 ft., and the number of holes fired simultaneously varying from 1 to 40. From this plant, we shipped oxygen long distances. In some shipments the oxygen was held in the transportation containers as long as 96 hr. For this purpose we used transportation containers of 50-liter capacity, which have a loss of about 3 per cent. in weight per 24 hr. at rest, and about twice this when agitated in transportation.

Cartridges

Most of the cartridges for this work were 4½ by 18 in. A few 7 by 24-in. catridges were used in blast No. 13. These cartridges were made of our regular blacks, but were made in cylindrical cotton bags, instead of with paper, as it was found that such large paper cartridges were too fragile to be handled.

Soaking Boxes

Because of the large size of cartridges and the necessity of soaking a great many at one time, we developed large soaking tanks which were merely copper-lined boxes insulated with hair felt.

Method of Loading Holes

A continuous length of cordeau from the bottom to the top of the hole was used for these large blasts. This cordeau was first placed in the hole, and all the cordeau from the various holes were spliced to a common trunk which was detonated electrically by a single cap.

In introducing the cartridges in the hole, a copper-flared funnel about 6 ft. long was used to guide the cartridge, and also to keep surface material from falling into the hole. The stemming used was generally loose earth merely shoveled into the hole without tamping, usually filling the hole to the collar. No difficulty was encountered with stemming blowing out.

There was also no difficulty in loading wet holes. We successfully blasted holes which were filled with water to the collar.

Table 16 gives a resume of the data on all of these blasts, forty-nine in number.

Figs. 36, 37 and 38 are photographs of blast No. 2, before blasting, after blasting, and after the steam shovel had cleaned out the material. These are typical of the quarry blasting with approximately 25 to 30-ft. benches.

Figs. 39 and 40 show blast No. 7 made in iron ore, 40-ft. bench. This shows a line of holes occupying a space along the bench of 500 ft., part of the blast being made with dynamite, and part with L. O. X. From the appearance of the material after blasting, no difference could be observed between the rock shot with dynamite, and that with L. O. X.

FIG. 36.—VIEW BEFORE LIQUID-OXYGEN BLAST. CALCITE QUARRY CO., MYERSTOWN, PA.

FIG. 37.—VIEW AFTER LIQUID-OXYGEN BLAST.

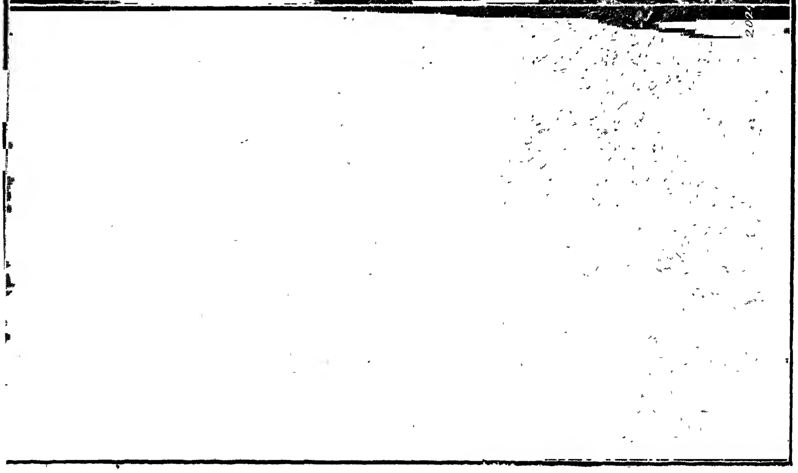


FIG. 38.—VIEW AFTER BLAST HAD BEEN CLEANED UP BY STEAM SHOVELS, SHOWING CLEAN BREAK.

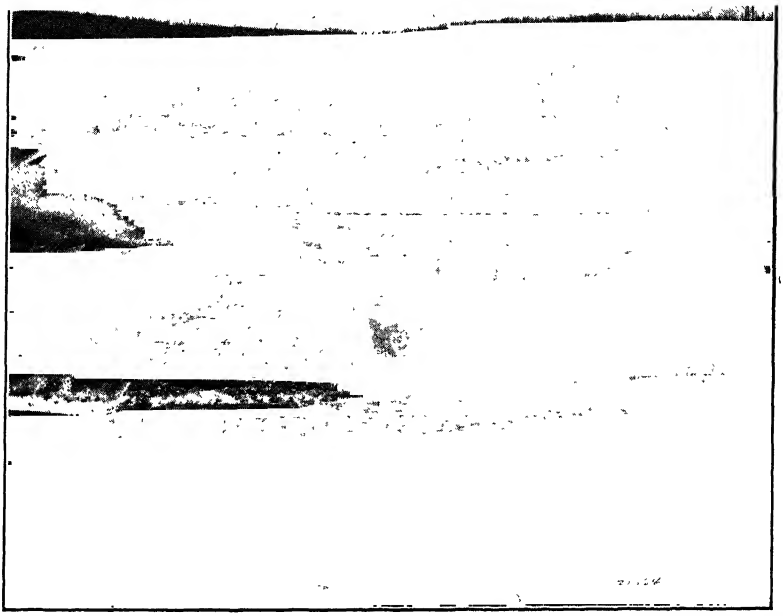


FIG. 39.—FORTY-FOOT BENCH OF IRON ORE BEFORE SHOOTING. LEFT END OF SHOT L. O. X.

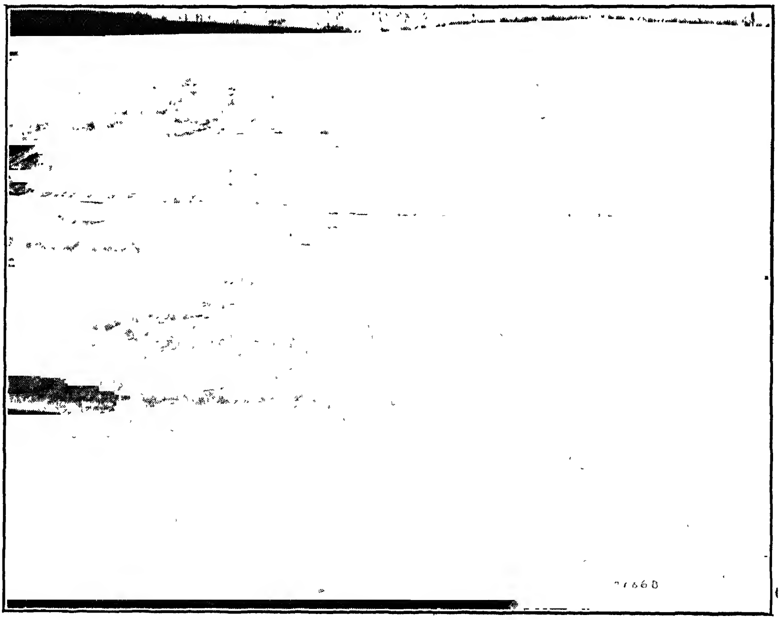


FIG. 40.—FORTY-FOOT BENCH OF IRON ORE AFTER SHOOTING.

In all these blasts, the amount of L. O. X. to be used was calculated in the following way. From the amount of dynamite commonly used, according to the quarry or mine officials, and from the estimated time needed for firing, the amount of L. O. X. needed was determined by using Figs. 20 and 25. As a check on this method, all shots, if added, would give a total pendulum swing of 382,812 in., for the dynamite that would have been used according to quarry practice, as compared with the calculated pendulum swing of 323,580 in. from the total amount of L. O. X. used. The actual result, therefore, was 18 per cent. better than the calculated strength. This increase in strength is probably due to the fact that large cartridges were used in these tests, whereas our data were compiled from results with small cartridges.

An investigation of shots Nos. 5, 22, 27 and 43, which were not successful, disclosed the fact that they were not properly charged or fired, due either to miscalculation or to inadequate facilities at the time of charging the holes. In shot No. 13, it was found that what was assumed to be broken material at the toe of the bench was not broken material, but solid toe left by a previous blast. For this reason, the shot was not successful. All of the other shots were successful in every respect.

Taking the total of the dynamite replaced, the amount of oxygen and dry cartridge used in all these blasts, it appears that 1 lb. of dynamite is replaced by 0.935 lb. of oxygen, measured as produced at the liquid oxygen plant. This, therefore, includes all losses of every sort. It also appears that 0.201 lb. of dry cartridges are required to replace 1 lb. of dynamite.

Cost

Table 4 gives the cost of cartridges. Table 1 gives the cost of oxygen.

To replace 1 lb. of dynamite for this class of service, therefore, requires 0.935 lb. of oxygen at $0.0229 per pound or $0.0214, and 0.201 lb. of cartridge at $0.14 per pound or $0.0283. This makes a total cost of $0.0494 to replace 1 lb. of dynamite.

Conclusions

1. L. O. X. is admirably adapted for quarry work and open-pit mining.
2. The cost of L. O. X. for this work is much less than dynamite.
3. The danger of digging into missed holes is eliminated.
4. There is no difficulty in handling the material or charging the holes.
5. The amount of labor needed to load holes with L. O. X. is less than with dynamite, as the cartridges are simply dropped into the holes and no ramming is necessary.
6. As there are no lights in open-pit work, there is no danger of the cartridges catching fire. Even should they do so from sparks from steam

shovels or locomotives, their burning in the open air would involve no serious consequences.

ACKNOWLEDGMENTS

The authors wish to thank the officials of the Witherbee-Sherman Co. for their earnest cooperation and interest, also the staff of the Pittsburgh laboratory of the Bureau of Mines for the use of its facilities for laboratory work, and the Calcite Quarry Co. for cooperation.

The authors also acknowledge their indebtedness to George B. Holderer of the Air Reduction Co., and to E. H. Dickenson of the Ingersoll-Rand Co., for their long and laborious laboratory and field work.

DISCUSSION

B. F. TILLSON, Franklin, N. J.—What percentage is the standard dynamite referred to in the table?

F. W. O'NEIL.—Forty per cent.

B. B. HOOD, Bridgeport, Conn.—Must the hole ventilated be kept ventilated?

F. W. O'NEIL.—There are enough cracks in the hole and the stemming does not fit closely enough to prevent the gas from leaking out. Occasionally in very tight ground with very tight stemming, enough pressure will build up inside to blow out the stemming. In Lorraine they devised a contrivance with which to measure the amount of pressure that could be developed in the drill hole; I think 1 atmos. was the maximum.

W. R. WRIGHT, Chicago, Ill.—Are the cartridges loaded on the job or at the plant?

F. W. O'NEIL.—In the quarry work they are loaded in a building near the quarry and then put in a dump cart. They are thrown in like so many sacks of sand and are dropped into the holes.

MEMBER.—What is the temperature and duration of the flame, compared with that of black blasting powder, for instance?

F. W. O'NEIL.—We have only seen the cartridges lighted in the open air and allowed to burn; they burn like the most beautiful Roman candles you ever saw. We took pictures underground using them as a light.

MEMBER.—Would its use be much more dangerous in a coal mine than black blasting powder?

F. W. O'NEIL.—We do not recommend it for coal mining; in fact, we say frankly that it should be kept out of the coal mine.

W. R. WRIGHT.—It is in a sense highly explosive, probably as strong as T.N.T.

F. W. O'Neil.—It is equivalent to 40 per cent. standard dynamite; it is not quite as powerful as 60 per cent.

G. C. Crossley, Trenton, N. J.—When the cartridges were placed in holes containing water, did they freeze the water?

F. W. O'Neil.—The water boils just as if the cartridges were hot; that is because the gas of the oxygen is passing through the water.

G. C. Crossley.—Would not the use of smaller cartridges and placing water in the hole make the charge more efficient by closing up the hole?

F. W. O'Neil.—Usually underground, with bottom holes, there is enough water to float the cartridge out.

W. R. Wright.—Does the operator, or plant owner, have to manufacture those cartridges or are they supplied?

F. W. O'Neil.—The cartridges can be made by anybody who will buy the necessary materials.

G. W. Farny, Morris Plains, N. J.—If you put a little mud at the bottom of a vertical hole, put in the tube that you discarded outside, put in the dry cartridge, and then poured in the liquid oxygen, what would happen?

F. W. O'Neil.—The first attempts to use liquid oxygen were by that system. European patents were taken out for various ways of pouring the liquid oxygen on the dry cartridge in the hole. But the plan is not satisfactory. It is too wasteful; as the cartridge is becoming impregnated, the liquid is also being evaporated.

B. F. Tillson.—What other limitations are there on the diamater of the cartridge?

F. W. O'Neil.—Underground, cartridges smaller than 1⅛ in., must be shot in 10 min. This is about the smallest size any one should contemplate using.

B. F. Tillson.—You do not think that 1 in. cartridges could be used?

F. W. O'Neil.—No; the life decreases rapidly with the size of the cartridge; the larger the cartridge, the smaller is the evaporating surface per unit of volume in it.

G. W. Farny.—Would you recommend the use of liquid oxygen on farms and for blasting out trees or trenches?

F. W. O'Neil.—No. When starting up, one of these liquid-oxygen plants must be run about 1½ hr. to establish the temperatures.

G. St. J. Perrott, Pittsburgh, Pa. (written discussion).—The explosive strength of L. O. X. cartridges, though obtained from comparatively

little data, agrees remarkably well with the results previously published.[4] The method of plotting explosive strength has been changed slightly, and we now plot oxygen content against the weight of standard dynamite equivalent to unit weight of L. O. X. cartridge, instead of using the deflection of the pendulum per unit weight of dynamite. This procedure makes calculation from the curves somewhat simpler. From the results of a large number of tests of cartridges of varying size, density and composition, average curves have been prepared from which the strength in terms of dynamite of any L. O. X. cartridge may be obtained from the data of an evaporation test.

Although the curve in Fig. 15 is a straight line the strength of cartridges of increasing density fired under identical oxygen percentages does not necessarily increase in the same proportion as the weight. Many experiments have shown that the explosive strength per unit weight for a cartridge of high density is less than for a cartridge of low density. Recent experiments have shown that under conditions in the borehole, equilibrium is not attained and not all of the carbon of the denser cartridges is consumed.

The curves in Fig. 28 show a decrease in strength for 5 by 18-in. cartridges at packing densities above 0.31 g. per cc. Experimental data obtained by the Bureau of Mines on 2-in. cartridges, show that for evaporation periods of 15 min., a cartridge of 0.38 g. per cc. shows maximum strength, and there is no reason to believe that this or a slightly higher density might not give optimum strength for certain evaporation periods in the case of the 5-in. cartridges.

Experiments[5] at Leadville, Colo.; Pachuca, Mexico; and Cerro de Pasco, Peru, have shown that considerable carbon monoxide may be produced by L. O. X. cartridges. A cartridge packed with gas black to a density of 0.30 g. per cc., fired after an evaporation period of 10 min., produces in actual blasting several times as much CO as an equivalent quantity of gelatin dynamite. Rounds fired when all the cartridges contain enough oxygen for complete combustion to CO_2, still produce some CO; only when firing time or density are so reduced is considerable excess of oxygen present at the time of firing so that the amount of CO produced falls to a negligible quantity. The good ventilation at Mineville probably prevented the Messrs. O'Neil and Van Fleet from detecting carbon monoxide, which was undoubtedly produced. In ground containing even a slight amount of pyrite, considerable SO_2 and some H_2S are formed.

[4] G. St. J. Perrott: Properties of Liquid Oxygen Explosives. *Trans.* (1925), **71**, 1248.

[5] G. St. J. Perrott: Underground Blasting in Metal Mines with Liquid Oxygen Explosives. *Eng. & Min. Jūl.-Pr.* (1926), **121**, 357.

Report of Sub-committee on Coal Mining to Committee on Ground Movement and Subsidence

(New York Meeting, February, 1926)

The Sub-committee on Coal Mining, since its appointment, has been collecting data about subsidences in coal mines in this country, and thinks that almost all of the available data about occurrences of this sort in bituminous mines has been collected. Similar data is being compiled for the anthracite regions, which it is hoped will be ready for the 1927 meeting.

As most of the bituminous seams are nearly level, the records obtained give the problem in its simplest form, without complications due to the pitch of the seam.

Copies of the questionnaire, Appendix A, were mailed to all of the coal operators' associations, to most of the large operating companies and to many engineers, with the request that any data on the subject be returned to the committee, or that advice of any subsidences known be given: the replies to most of the questionnaires were general, although some specific cases were given. Many of the companies were unable to reply to all questions, as they lacked the necessary data, but in nearly every case where accurate data was available, it was cheerfully furnished, and any additional information requested was secured. All specific cases of subsidence were followed up by correspondence, and in several cases by personal inspection. Figs. 1 to 31 were made from the information furnished. Table 1 gives measurements of elevations taken in one case at intervals from October, 1913, to May, 1918.

With one exception, where the evidence cannot be released on account of a lawsuit, it is believed that the cases submitted with this report include practically all that have happened in bituminous coal mining in this country of which actual measurements were secured. Some of these have been published by Young and Stoek,[1] and some have been secured from other publications. In collecting these data, the sub-committee had the Engineering Societies Library extend its bibliography to date; this list is given in Appendix B.

To assist in the study of the problem, all of the subsidence literature in English was examined for cases in which specific measurements were given, and the essential data for each case are shown in Appendix C.

[1] L. E. Young and H. H. Stoek: Subsidence Resulting from Mining. Univ. Ill. Eng. Exp. Sta. *Bull.* 91 (1916), 179 pp.

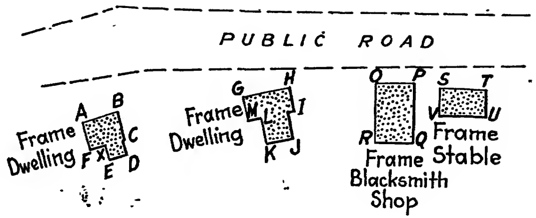


TABLE I.—*Elevations Taken on Buildings over a Coal Mine at Intervals from Oct. 20, 1913 to May 23, 1918*

Monuments	1913 Oct. 20	1913 Nov. 14	1913 Dec. 22	1914 Jan. 20	1914 Feb. 21	1914 Mar. 28	1914 April 27	1914 May 23	1914 June 30
A	1250 26	1250 21	1250 20	1250 18	1250 22	1250 17	1250 20	1250 18	1250 19
B	1250 37	1250 31	1250 30	1250 30	1250 32	1250 28	1250 30	1250 27	1250 29
C	1250 21	1250 15	1250 15	1250 13	1250 18	1250 10	1250 14	1250 12	1250 12
D	1248 48	1248 40	1248 40	1248 40	1248 46	1248 37	1248 40	1248 37	1248 39
E	1248 83	1248 74	1248 77	1248 73	1248 75	1248 66	1248 68	1248 66	1248 57
X	1250 61	1250 54	1250 54	1250 53	1250 57	1250 52	1250 54	1250 52	1250 52
F	1249 89	1249 84	1249 85	1249 83	1249 85	1249 80	1249 82	1249 81	1249 83
G	1250 95	1250 87	1250 89	1250 88	1250 90	1250 86	1250 90	1250 87	1250 87
H	1251 01	1250 92	1250 93	1250 93	1250 97	1250 91	1250 92	1250 92	1250 92
I	1250 91	1250 84	1250 84	1250 82	1250 87	1250 81	1250 80	1250 82	1250 82
J	1250 71	1250 62	1250 63	1250 62	1250 66	1250 62	1250 64	1250 60	1250 60
K	1250 70	1250 60	1250 60	1250 63	1250 62	1250 58	1250 62	1250 60	1250 61
L	1251 09	1250 99	1251 00	1251 00	1251 02	1250 98	1251 02	1250 99	1250 99
M	1250 75	1250 66	1250 66	1250 67	1250 70	1250 66	1250 68	1250 67	1250 68
O	1256 48	1256 39	1256 41	1256 40	1256 42	1256 39	1256 38	1256 38	1245 99
P	1246 39	1246 28	1246 30	1246 30	1246 34	1246 28	1246 30	1246 27	1246 29
Q	1245 74	1245 64	1245 65	1245 65	1245 67	1245 61	1245 65	1245 62	1245 63
R	1245 56	1245 42	1245 46	1245 45	1245 46	1245 40	1245 45	1245 42	1245 42
S	1244 42	1244 32	1244 34	1244 37	1244 37	1244 32	1244 34	1244 32	1244 28
T	1244 45	1244 34	1244 36	1244 37	1244 37	1244 36	1244 37	1244 34	1244 36
U	1244 21	1244 10	1244 13	1244 12	1244 12	1244 10	1244 13	1244 09	1244 09
V	1244 16	1244 04	1244 07	1244 04	1244 06	1244 03	1244 05	1244 02	1244 00

Monuments	1914 July 16	1914 Aug. 24	1914 Sept. 17	1914 Oct. 14	1914 Nov. 16	1914 Dec. 14	1915 Jan. 15	1915 Feb. 12	1915 Mar. 20	1915 April 19	1916 June 15	1918 May 23	Monuments
A	1250 24	1250 19	1250 17	1250 13	1250 22	1250 20	1250 20	1250 23	1250 18	1250 15	1250 01	1249 63	A
B	1250 34	1250 30	1250 28	1250 24	1250 32	1250 28	1250 29	1250 33	1250 29	1250 26	1250 16	1249 76	B
C	1250 16	1250 10	1250 11	1250 09	1250 15	1250 16	1250 13	1250 16	1250 11	1250 08	1250 01	1249 72	C
D	1248 43	1248 38	1248 36	1248 34	1248 43	1248 37	1248 34	1248 42	1248 34	1248 36	1248 21	1247 97	D
E	1248 68	1248 63	1248 64	1248 59	1248 69	1248 68	1248 65	1248 67	1248 64	1248 62	1248 52	1248 27	E
X	1250 57	1250 52	1250 52	1250 47	1250 57	1250 53	1250 54	1250 57	1250 52	1250 50	1250 36	1250 10	X
F	1249 88	1249 83	1249 82	1249 77	1249 85	1249 85	1249 84	1249 86	1249 81	1249 78	1249 65	1249 37	F
G	1250 89	1250 86	1250 85	1250 82	1250 89	1250 89	1250 86	1250 89	1250 85	1250 79	1250 78	1250 27	G
H	1250 96	1250 95	1250 93	1250 88	1250 95	1250 96	1250 94	1250 97	1250 92	1250 89	1250 84	1250 23	H
I	1250 86	1250 82	1250 82	1250 78	1250 85	1250 86	1250 84	1250 87	1250 82	1250 81	1250 74	1250 23	I
J	1250 65	1250 61	1250 62	1250 58	1250 64	1250 63	1250 63	1250 66	1250 60	1250 59	1250 54	1250 15	J
K	1250 65	1250 59	1250 60	1250 57	1250 62	1250 60	1250 61	1250 65	1250 58	1250 57	1250 52	1250 19	K
L	1251 04	1250 98	1250 99	1250 96	1251 02	1251 01	1251 00	1251 03	1250 98	1250 95	1250 91	1250 48	L
M	1250 71	1250 64	1250 66	1250 62	1250 69	1250 68	1250 67	1250 70	1250 66	1250 62	1250 58	1250 15	M
O	1246 02	1246 02	1246 00	1245 97	1246 00	1246 02	1246 04	1246 02	1246 06	1246 01	1246 02	1245 11	O
P	1246 30	1246 29	1246 27	1246 24	1246 29	1246 31	1246 29	1246 31	1246 28	1246 26	1246 15	1245 24	P
Q	1245 67	1245 61	1245 61	1245 59	1245 63	1245 66	1245 65	1245 66	1245 59	1245 57	1245 55	1244 82	Q
R	1245 46	1245 4[illegible]	1245 41	1245 37	1245 43	1245 44	1245 44	1245 46	1245 39	1245 38	1245 35	1244 78	R
S	1244 30	1244 30	1244 28	1244 25	1244 32	1244 38	1244 28	1244 34	1244 27	1244 24	1244 25		S
T	1244 38	1244 37	1244 35	1244 31	1244 37	1244 35	1244 35	1244 37	1244 35	1244 33	1244 33		T
U	1244 15	1244 08	1244 10	1244 07	1244 13	1244 14	1244 12	1244 15	1244 09	1244 07	1244 07		U
V	1244 08	1244 01	1244 03	1243 99	1244 05	1244 04	1244 05	1244 08	1244 01	1243 99	1243 99		V

Many more cases were mentioned in the papers searched, but no specific data of value to us were given. Many cases are mentioned in German, Belgian, French and Austrian publications, but no translations of these are available.

The information about American cases was compiled in the same manner as for the foreign ones and is shown in Appendix D. In all cases reference is made to the authority for the data given, and to the print on which it is shown. As most of the companies and individuals requested that the identity of the cases reported be not divulged, no acknowledgments are possible, but the thanks of the committee are due the entire industry for the spirit of coöperation shown.

Deductions from Data at Hand

Until all data available, both here and abroad, have been collected and published, it is too soon to draw definite conclusions, but study of the data submitted with this, and the papers and discussions at the February 1923 meeting, appear to the writer to lead to certain deductions, which may have to be modified where all of the evidence is at hand.

The effects of subsidence may be considered under the following heads:

1. Those due to squeezes, or subsidence where the pillars have not been withdrawn.
2. Those in upper seams, due to mining seams beneath them.
3. Those due to mining by the room and pillar system.
4. Those due to mining by the longwall system.

Class 1—Subsidence Cases

All of the cases under Class 1, of which we have definite data, are in this country and mainly in Illinois and Oklahoma, where pillar withdrawal is seldom practiced. These cases, Nos. 1 to 12, 21, 28 to 32, 34 and 40 in Appendix D show that where pillars have not been drawn but where, under the areas affected, from 49 per cent. to 79 per cent. of the coal has been mined by room workings, that the surface has been cracked where the coal is 720 ft. deep; that a pond was formed, due to surface subsidence, where the coal was 625 ft. deep; and that surface subsidence as high as 79 per cent. of the seam thickness happened, in one case, where the seam was 487 ft. deep.

In every one of these cases, excepting No. 32, the seam was practically level and the subsidence was over the area mined out. In case No. 32, the seam dipped at an angle of 28°, and while the extreme crack was ahead of a vertical line from the face, it was back of a normal line to the seam.

In many of the Illinois mines the bottom is a fire clay, which will not usually stand the same pressure as a slate bottom, particularly where

water is present. In No. 40, are shown some sections where squeezes occurred, partly due to thick fireclay bottom, where less coal was mined than in sections with hard bottom which did not squeeze.

Class 2

Under Class 2, there are cases both here and abroad. Since the publication of the writer's paper[2] additional evidence on the effects of such mining has been obtained. No. 46, Appendix D, shows one case in Central Pennsylvania where two upper seams were badly disturbed by mining a lower one; the top seam, however, was being successfully mined but with an increased cost for timber and slate work, and an increased production of fine coal. Nos. 50, 51, 52 and 53 are cases where the thick Pittsburgh seam was mined many years ago and where overlying seams are now being mined. No. 51 is the only case we found, except probably No. 46, where the upper seam has been cut out entirely by its subsidence over worked out areas, although hundreds of places have been worked in such conditions. As in all of these cases the extraction in the lower seam was irregular and incomplete, such a condition is remarkable.

In view of the additional evidence, and the discussion of the paper mentioned at the February, 1923, meeting, the conclusions stated then should be slightly modified, as follows:

1. Mining an upper seam after a lower one has been removed can almost always be successfully done when the thickness of the intervening strata is 19 ft. or more, the thickness of coal mined in the lower seam not exceeding 8 to 9 ft.

2. The lower seam should be entirely removed from any area, and time should be allowed for settlement, before work is started in an upper seam; the more complete and uniform the extraction, the less will be the likelihood of trouble in the upper seam.

3. Working in an upper seam should not be attempted while pillars are being extracted in a seam below it.

4. If the coal in the bottom seam has been mined, as in No. 2, the percentage of recovery in an upper seam will not be materially reduced, nor the cost or danger of mining greatly increased, because a lower seam has been mined. Even if the mining of the lower seam has been imperfectly done, an upper seam can almost always be successfully mined, but with a somewhat greater expense and a larger production of fine coal.

Class 3

In Class 3 are subsidences due to mining by the room and pillar system. There are records of such cases both abroad and in this country.

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[2] Howard N. Eavenson: Mining an Upper Bituminous Seam after a Lower Seam has been Extracted. *Trans.* (1923) **69**, 398.

Nos. 35, 36 and 37, Appendix C, show some English cases; Nos. 12, 13, 14 and 15 show some East Indian cases; No. 43 is a case from South Africa; and nearly all of those in Appendix D, are from this type of mining.

Nos. 36 and 37 in Appendix C, and Nos. 20, 32, 39, 41 and 55 in Appendix D, are the only cases in this type of mining where the effects of surface subsidence are shown over the solid coal; in every other case, and in a great number quoted in publications, but for which no specific data were given, the subsidence was over the mined area. The practical universality of this condition indicates that the cases mentioned may be exceptions to the general rule, and that there may be circumstances connected with them that make the overlying cover subside differently from its usual action over this type of mining.

In No. 20, Appendix D, Note 1 explains the circumstances of pillar removal around this area. All of the coal on two sides of the triangle had been removed, but not enough on the other side, at the same time to make a fall. It is probable that the strata on the two sides were disturbed when pillar withdrawal was finally begun on the third side, and this disturbance caused the subsidence over the solid coal.

While in No. 32, Appendix D, the limit of subsidence was over the solid coal, ahead of a vertical line from the face, it was behind a normal line to the face.

In No. 39, Appendix D, the limit of subsidence was over the solid coal, the house where it was observed being on one side of a property line on the opposite side of which all of the coal had been removed. It is probable that the unusual thickness of the surface soil and wash here influenced the extent of the surface subsidence considerably.

In No. 41, Appendix D, the cracks appeared over solid coal. It is possible that the removal of the coal at one side of the area affected left the strata in condition to be disturbed by the pillar withdrawal, coming directly uphill. This case has been the only one in the experience of this company where subsidence has occurred over the solid coal, but it has always noted the fact that the effects of subsidence are much more noticeable when the pillars are being drawn uphill, than when being mined in the opposite direction.

In No. 55, Appendix D, the surface was broken over solid coal for a distance of about one-fifth of the length of the pillar line.

Class 4

Class 4 consists of subsidences due to mining by the longwall system. While some longwall mining has been, and is being done in this country, no specific information is available about subsidences caused by it. George A Rice[3] says his observations of subsidences caused by longwall

[3] G. S. Rice: Some Problems in Ground Movement and Subsidence. *Trans.* (1923) **69**, 374.

mining in Northern Illinois show that the surface subsidence always extends beyond the area of excavation. No evidence about this point is given in the bulletin of the Illinois State Geological Survey.[4] The results of the effect of the "V" system of mining so far observed are shown at No. 56, Appendix D. Although the data obtained are not sufficient to allow any conclusions to be drawn, they show that up to date the subsidence has not extended beyond area excavated.

Experience in Great Britain has clearly established the fact that in every case where measurements of the effects of longwall mining have been made, the subsidence has extended beyond the area excavated and the writer understands, from the references made to European experiences, that this fact is also clearly demonstrated there.

It is evident from the data available that the method of mine working is one of the most important factors, if not the most important, in determining the effect upon the surface made by subsidence caused by bituminous coal mining.

Appendix E gives actual measurement of a fall in the thick Pittsburgh seam in the Georges Creek region, which extended up to the Redstone seam, a height of 48 ft. The positions of the inside of the fallen strata are only approximate, but they were located by actual measurement around the edges and on the top of the fall. This section is the highest one, actually measured, of which the writer has any knowledge, and it illustrates what may but which seldom does happen to an overlying seam.

Appendix F shows the writer's conception of the usual action of the overlying strata over an area from which all of the pillars have been withdrawn, based on the evidence submitted with this report. In view of the fact that in many modern mines the line of pillar withdrawal is frequently from 1500 to 4500 ft. long, and sometimes longer, it is difficult to believe that much support is derived from the strata at the sides of the excavated area, and it would seem that such pillar withdrawal is really a modified type of longwall.

It seems clear to the writer that this difference in results is almost entirely due to the fact that in genuine longwall workings the roof strata bend, and do not break, and consequently settle almost as a whole, while in the room and pillar system, a fall of strata is made at intervals, sometimes at distances as little as 10 or 15 ft. apart, but each fall is usually distinct and an appreciable time elapses before the next one. This difference in movement allows the strata over the room and pillar system to loosen up to some extent, and not to move in such a compact mass as the cover over longwall workings evidently does.

From the evidence given it is almost inconceivable to the writer that any subsidence over room and pillar workings could show to the

[4] L. E. Young: Surface Subsidence in Illinois Resulting from Coal Mining. Ill. State Geol. Survey Cooperative Coal Mining Ser., *Bull.* 17 (1916), 100 pp.

extent of case No. 49, Appendix C, where the extraction of a seam 4 ft. thick by longwall in England, resulted in a measured subsidence of the surface of 1.44 ft., or 36 per cent. of the seam thickness, at a depth of 2970 ft.

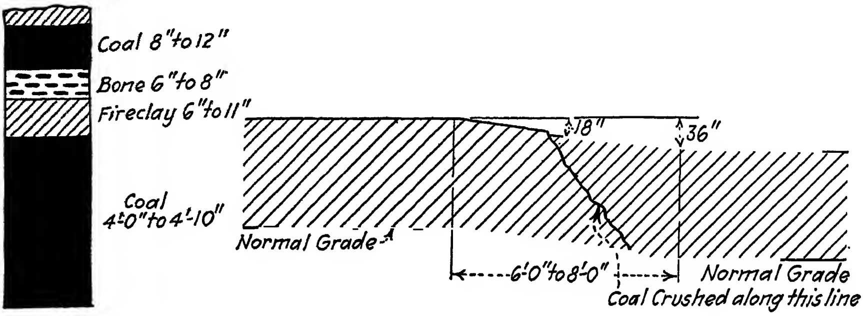


Fig. 1.—Break in Redstone seam when passing from solid coal to robbed area in the Pittsburgh seam.

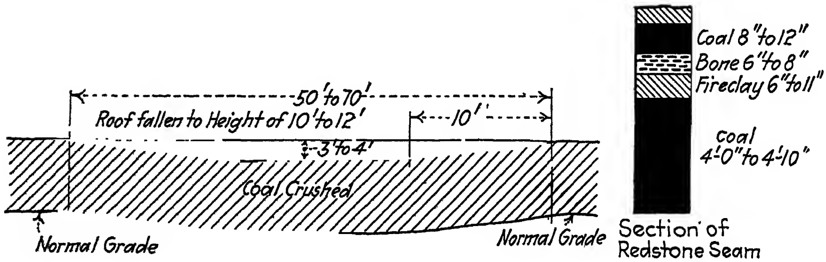


Fig. 2.—Another break in the Redstone seam when passing from solid coal to robbed area in the Pittsburgh seam.

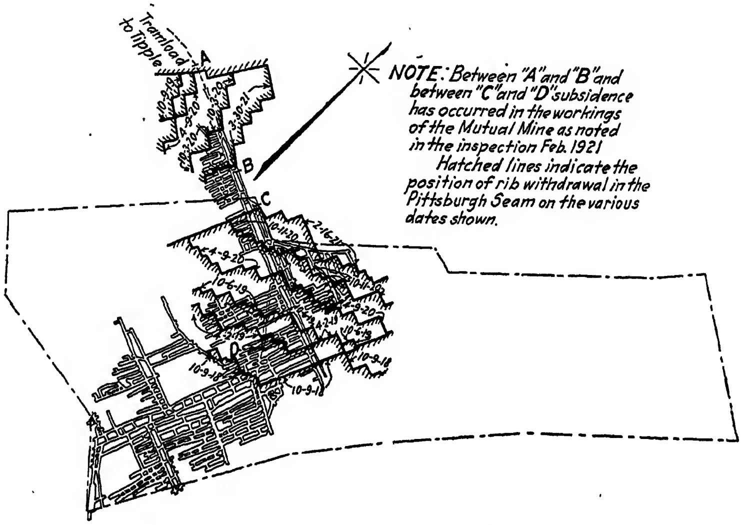


FIG. 3.—WORKINGS IN SEWICKLEY SEAM OVER ROBBED AREAS IN PITTSBURGH SEAM, FAYETTE CO., PA. BETWEEN A AND B AND BETWEEN C AND D SUBSIDENCE OCCURRED IN WORKINGS OF MINE, AS NOTED IN INSPECTION, FEB., 1921; HATCHED LINES INDICATE POSITION OF RIB WITHDRAWAL IN PITTSBURGH SEAM ON DATES SHOWN.

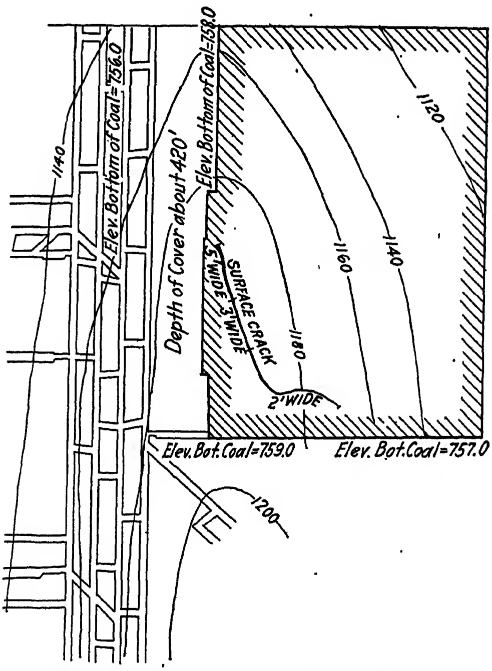


FIG. 4.—ANOTHER EXAMPLE OF SUBSIDENCE IN A COAL MINE IN FAYETTE CO., PA.

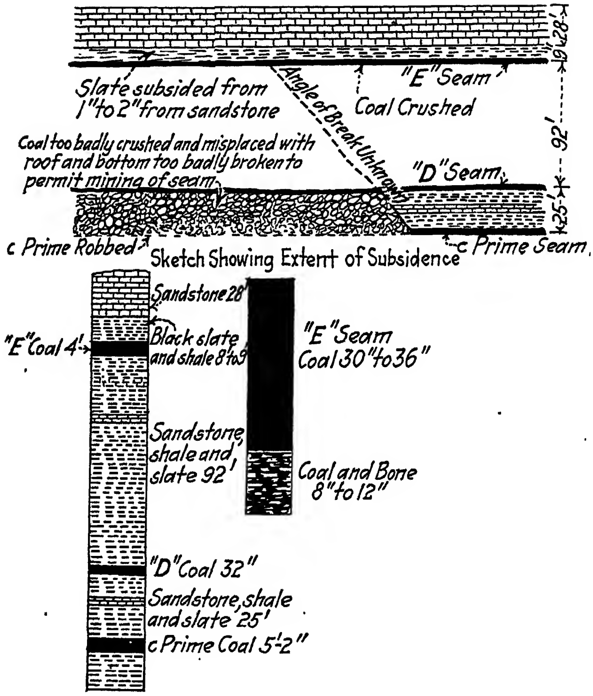


FIG. 5.—COLUMNAR SECTION OF STRATA AND CROSS-SECTION OF SEAMS AT A MINE NEAR JOHNSTOWN, PA.

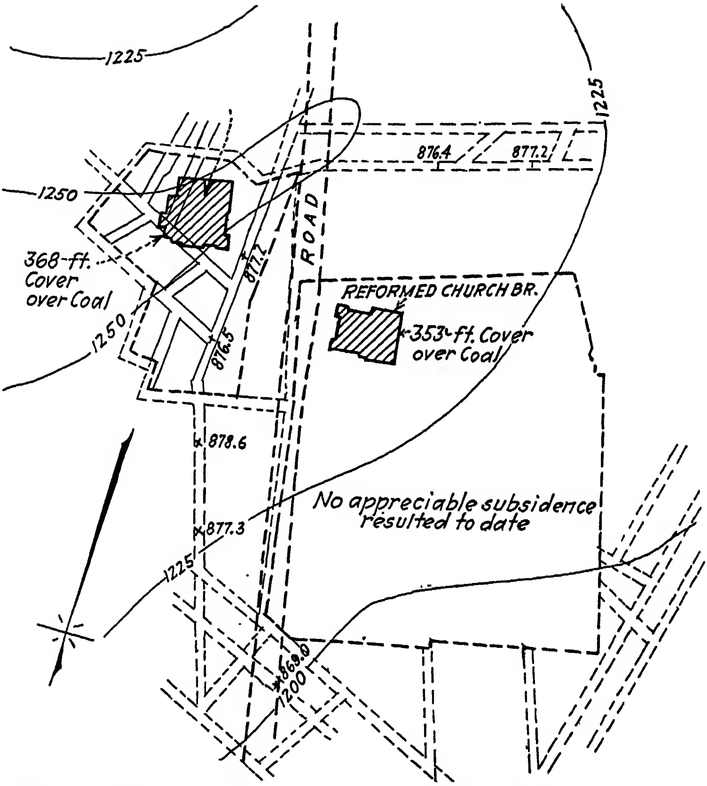


FIG. 6.—MAP OF MINED OUT AREA AROUND BUILDINGS OVER MINES IN SOUTHWESTERN PENNSYLVANIA.

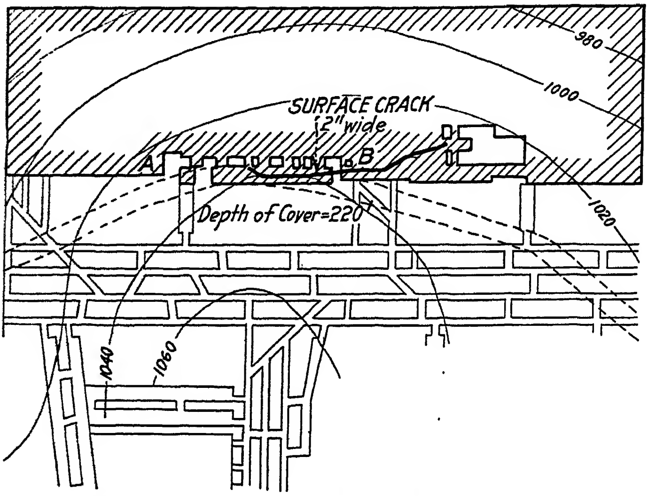


FIG. 7.—EXAMPLE OF SUBSIDENCE IN A COAL MINE, ALLEGHENY CO., PA.

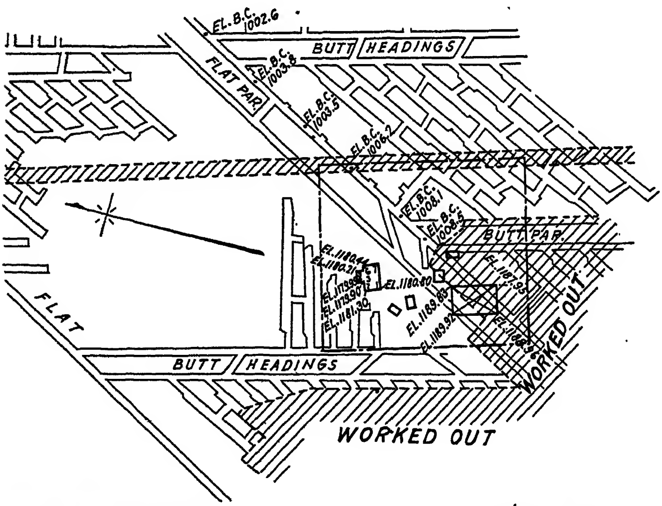


FIG. 8.—EXAMPLE OF SUBSIDENCE IN CONNELLSVILLE REGION, WESTMORELAND CO., PA.

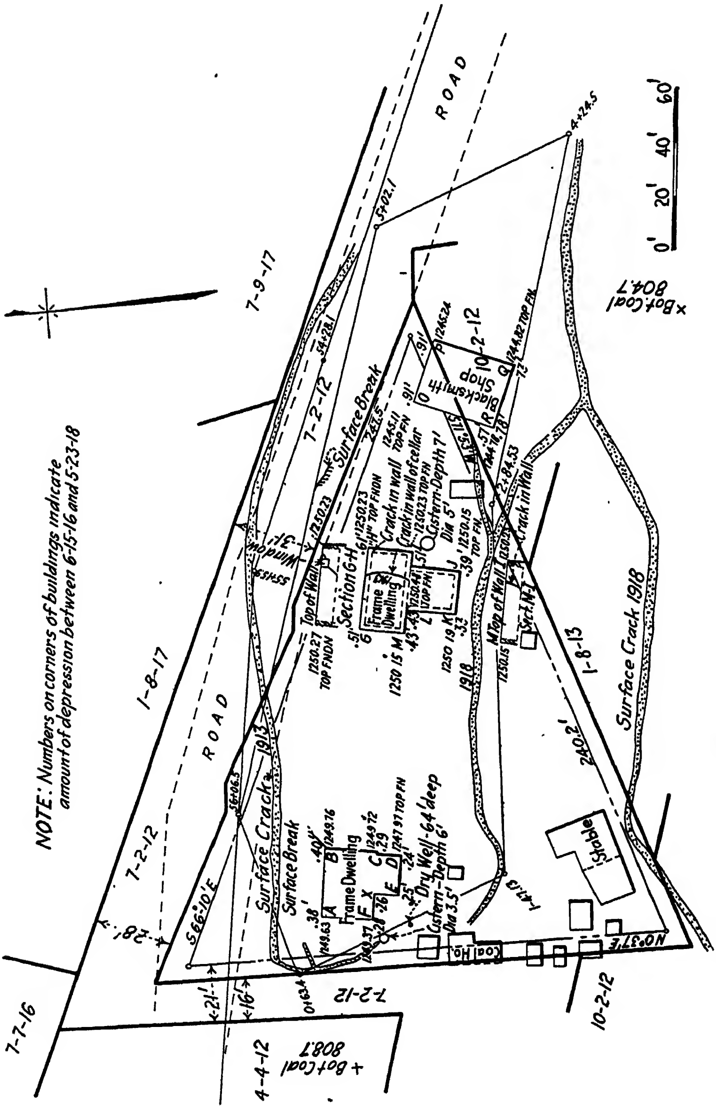


FIG. 9.—ANOTHER EXAMPLE OF SUBSIDENCE OF BUILDINGS, WESTMORELAND CO., PA.

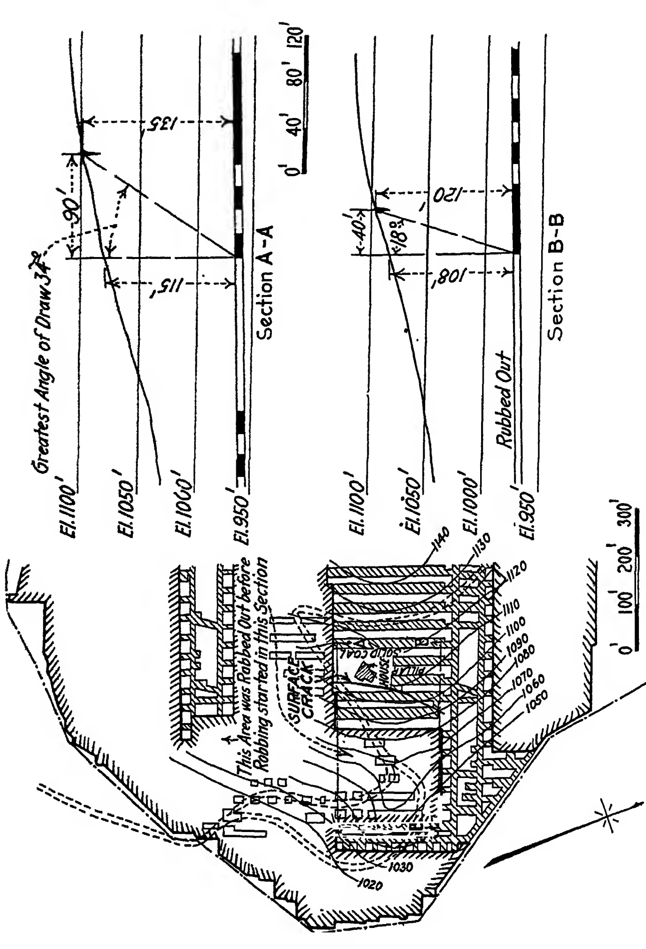


Fig. 10.—Example of subsidence in a coal mine in Washington Co., western Pennsylvania.

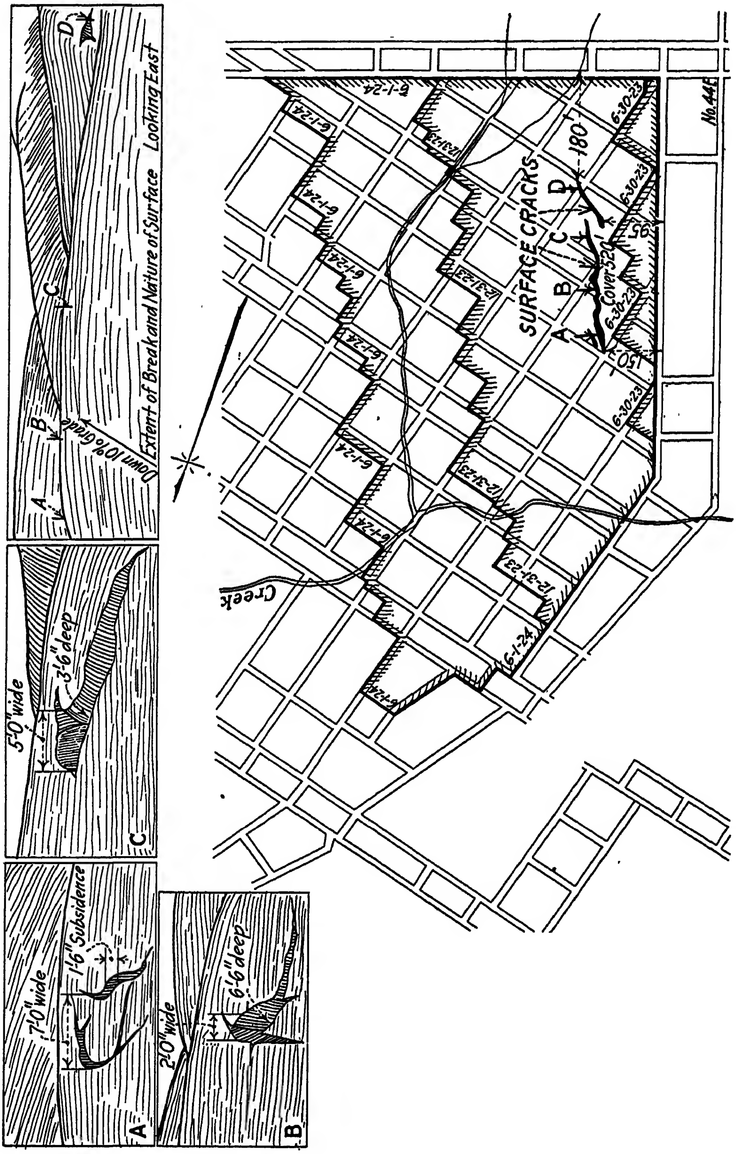


Fig. 11.—Example of subsidence in a coal mine, Greene Co., Pa.

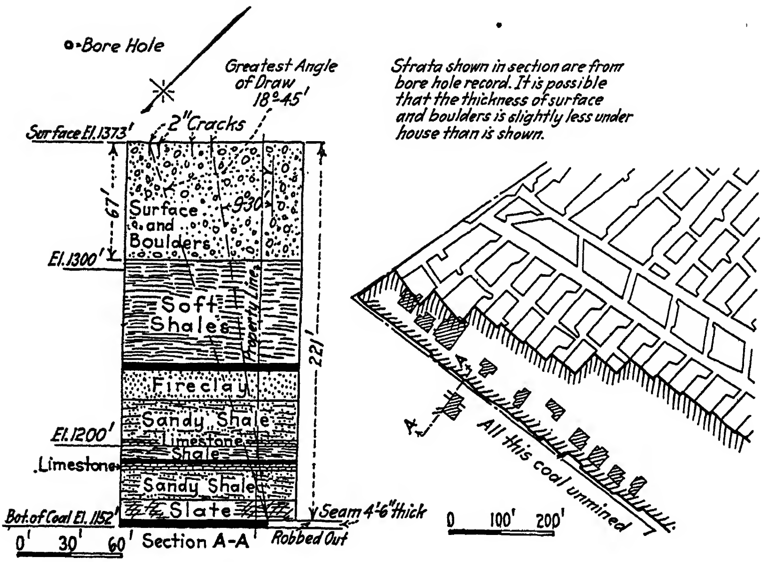


FIG. 12.—EXAMPLE OF SUBSIDENCE IN A COAL MINE, CAMBRIA CO., PA. STRATA SHOWN IN SECTION ARE FROM BORE-HOLE RECORD. IT IS POSSIBLE THAT THE THICKNESS OF SURFACE AND BOWLDERS IS SLIGHTLY LESS UNDER THE HOUSE THAN SHOWN HERE.

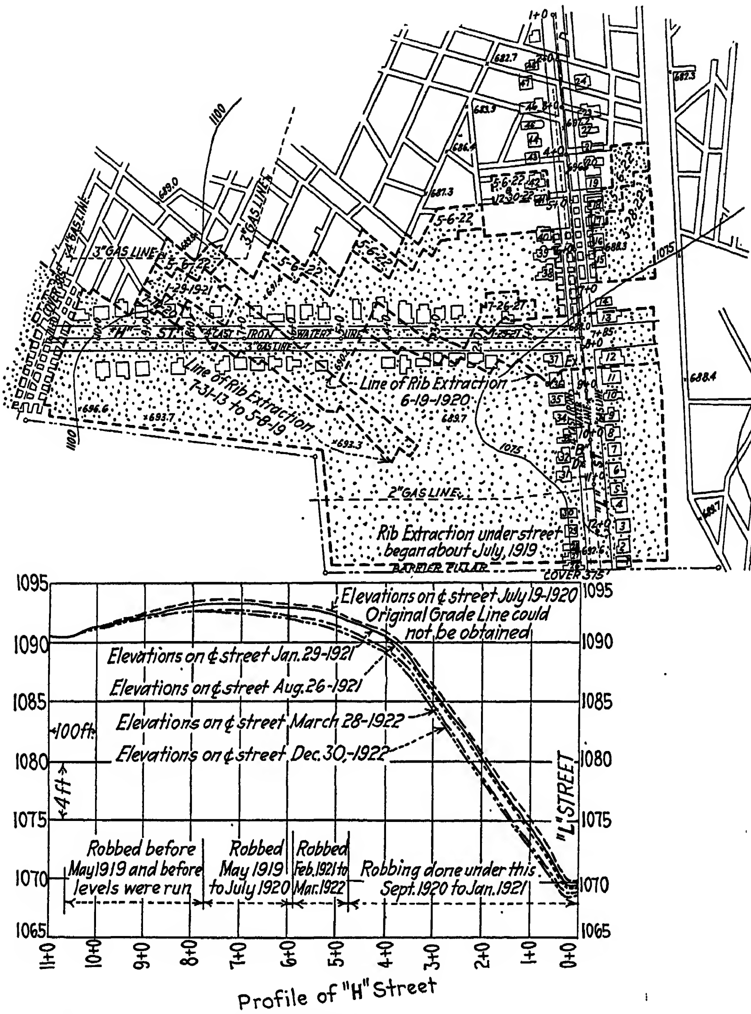


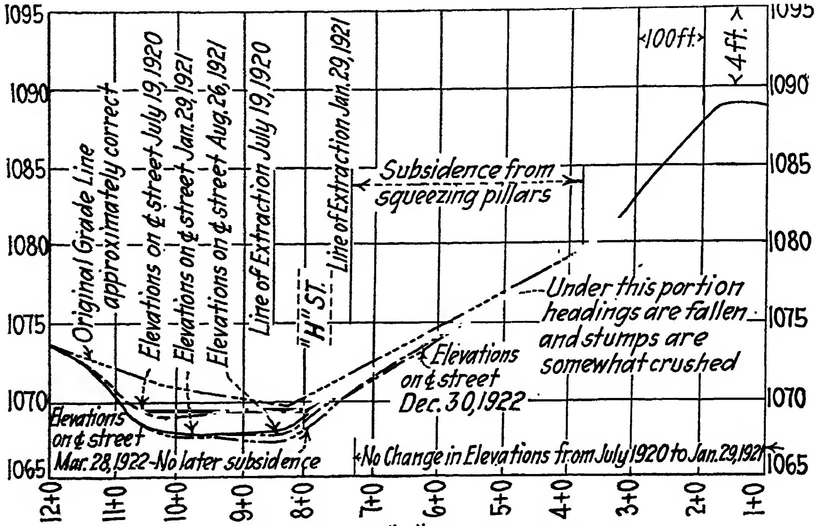
FIG. 13.—MAP SHOWING SUBSIDENCE ALONG STREETS IN A WESTERN PENNSYLVANIA TOWN.

At point *A* on H street the bell joints in the water line were drawn apart. At point *B* on L street, the ½-in. water line to house No. 32 was broken at the connection to the main. At point *C* on L street the 2-in. gas line was broken at the connection to the 1-in. line.

New sewer drops had to be placed at *D* and *E* to correspond with the new low points on L street. The 12-in. sewer line under L and H streets was not damaged. Streets H and L are paved with brick. A crack appeared at station 11 + 50 on L street but has now closed up and the road surfaces on both streets are solid.

The curbing consists of sandstone blocks 5 in. square varying in length from 3½ to 6 ft. These blocks have been thrown out of line for a distance of about 75 ft. in front of houses Nos. 5 and 6 on L street. One of these blocks is broken in half.

Houses No. 13 to 24 inclusive and 40 to 48 inclusive do not show any signs of settlement. Nos. 38 and 39 show 1-in. cracks around the moulding and beneath the mantel supports. With the exception of Nos. 5 and 6 the remaining houses show only slight indications of settlement. The porch supports on No. 6 are 1-in. out of line and the walls and ceilings are cracked. A 1-in. crack extends from the top to the bottom of the 2-foot stone foundation of house No. 5. The walls and ceilings are also cracked.



Profile of "L" Street

FIG. 13.—(*Continued.*)

DATA ON STREET SUBSIDENCE OVER MINED OUT AREA H STREET

Station of Survey	Original Cover, Feet	Subsidence after Robbing Was Done, Feet				Thickness of Seam, Per Cent.
		3 Mos.	10 Mos.	17 Mos.	26 Mos.	
0 + 0	382	0.4	0.6	1.1	1.3	16.2
1 + 0	386	0.6	1.0	2.1	2.4	30.0
2 + 0	391	0.4	0.7	2.0	2.3	28.7
3 + 0	399	0.4	0.6	1.7	2.0	25.0
4 + 0	404	0.3	0.5	1.4	1.7	21.2
5 + 0	402	0.4	0.4	1.3	1.5	18.7
6 + 0	402	0.4	0.4	1.2	1.4	17.5
7 + 0	402	0.4	0.4	1.2	1.3	16.2
8 + 0	401	0.4	0.4	1.4	0.9	11.2
9 + 0	399	0.2	0.2	0.2	0.5	6.2
10 + 0						
11 + 0						

DATA ON STREET SUBSIDENCE OVER MINED OUT AREA L STREET

Station of Survey	Original Cover, Feet	Subsidence after Robbing Was Done, Feet				Thickness of Seam, Per Cent.
		12 Mos.	18 Mos.	24 Mos.	38 Mos.	
4 + 0	382	Subsidence from squeezing pillars				
5 + 0	386					
6 + 0	387					
7 + 0	384					
7 + 50	383	0.7	0.9	1.0	1.4	17.5
8 + 0	382	0.6	1.2	1.4	2.0	25.0
8 + 50	381	0.5	1.8	2.1	2.5	31.2
9 + 0	381	0.7	2.2	2.4	2.7	33.7
9 + 50	380	1.2	2.7		2.8	31.2
10 + 0	381	1.9	3.0		3.1	38.7
10 + 50	381	2.1	2.9		3.1	38.7
11 + 0	381	1.3	1.7			

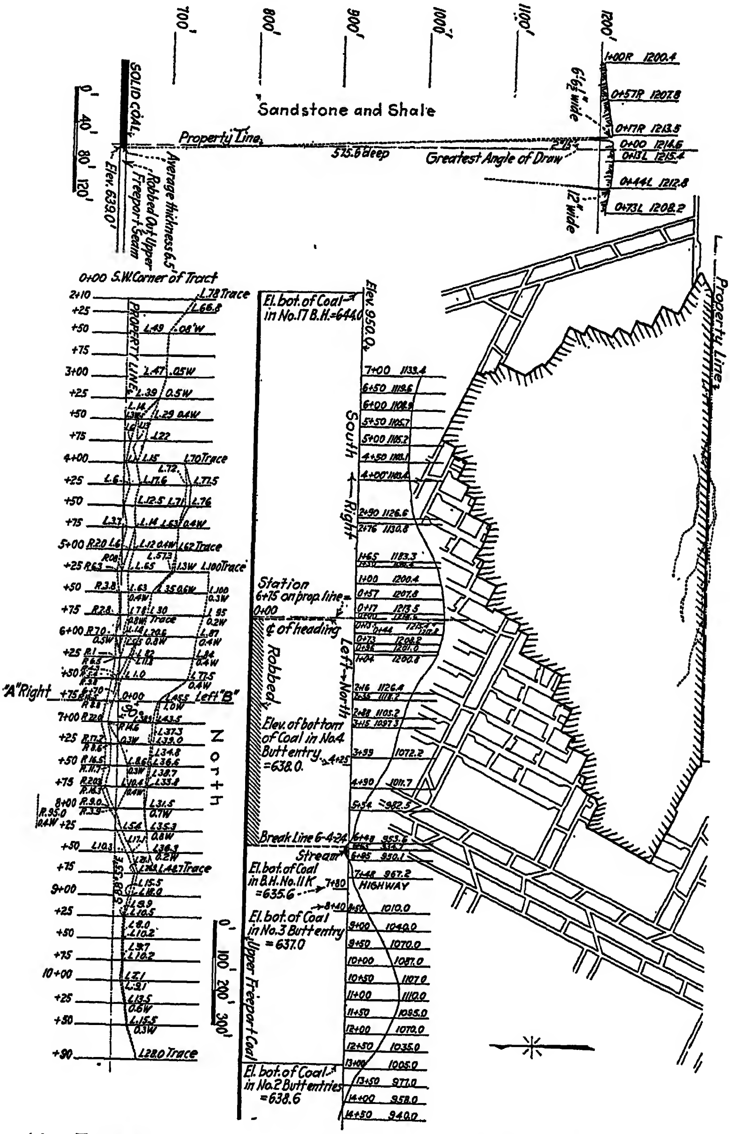


FIG. 14.—EXAMPLE OF SUBSIDENCE IN A COAL MINE IN ALLEGHENY CO., PA. ROBBING STARTED SEPT., 1922. FULL ROBBING LINE WAS OBTAINED DEC., 1922. SURFACE BREAKS WERE NOTICED EARLY IN 1923.

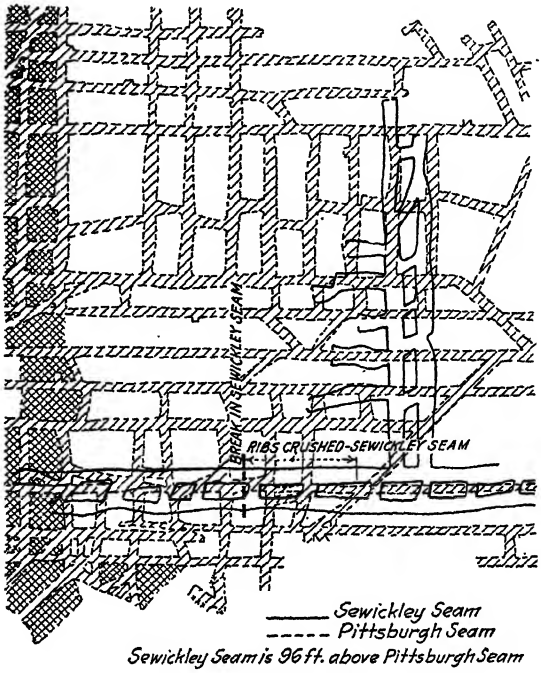


FIG. 15.—EXAMPLE OF SUBSIDENCE IN A MARYLAND COAL MINE.

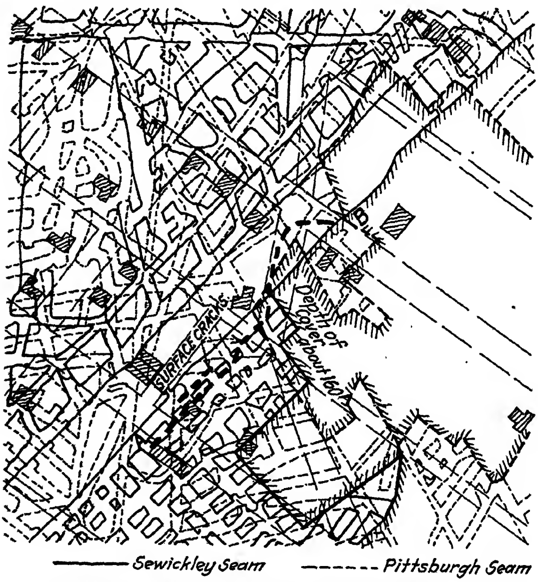


FIG. 16.—ANOTHER EXAMPLE OF SUBSIDENCE IN A MARYLAND COAL MINE.

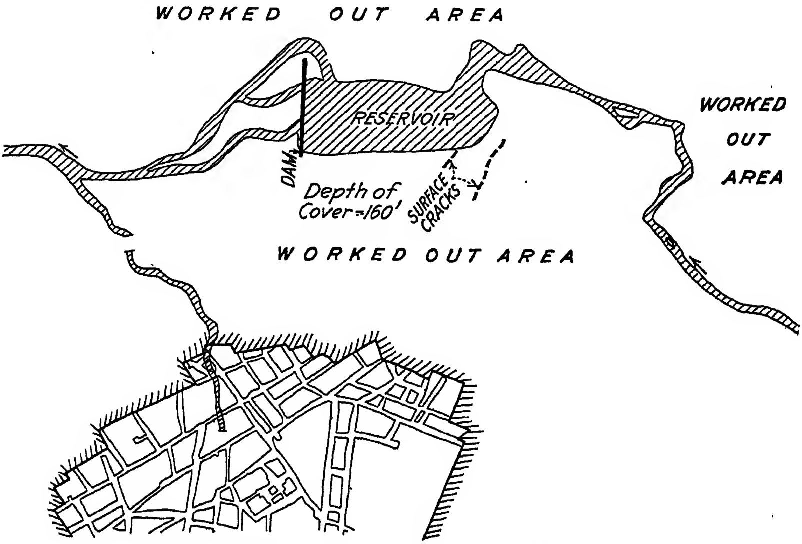
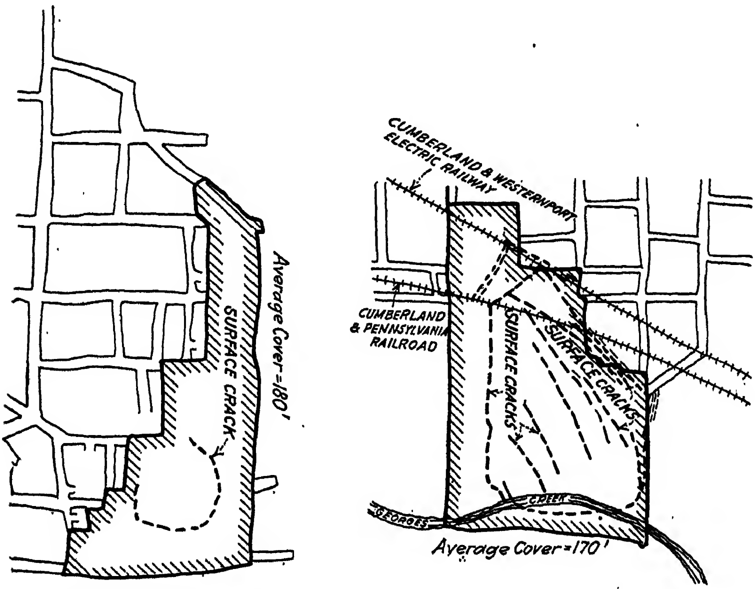


FIG. 17.—THIRD EXAMPLE OF SUBSIDENCE IN A MARYLAND COAL MINE.



18.—OTHER EXAMPLES OF SUBSIDENCE IN MARYLAND COAL MINES.

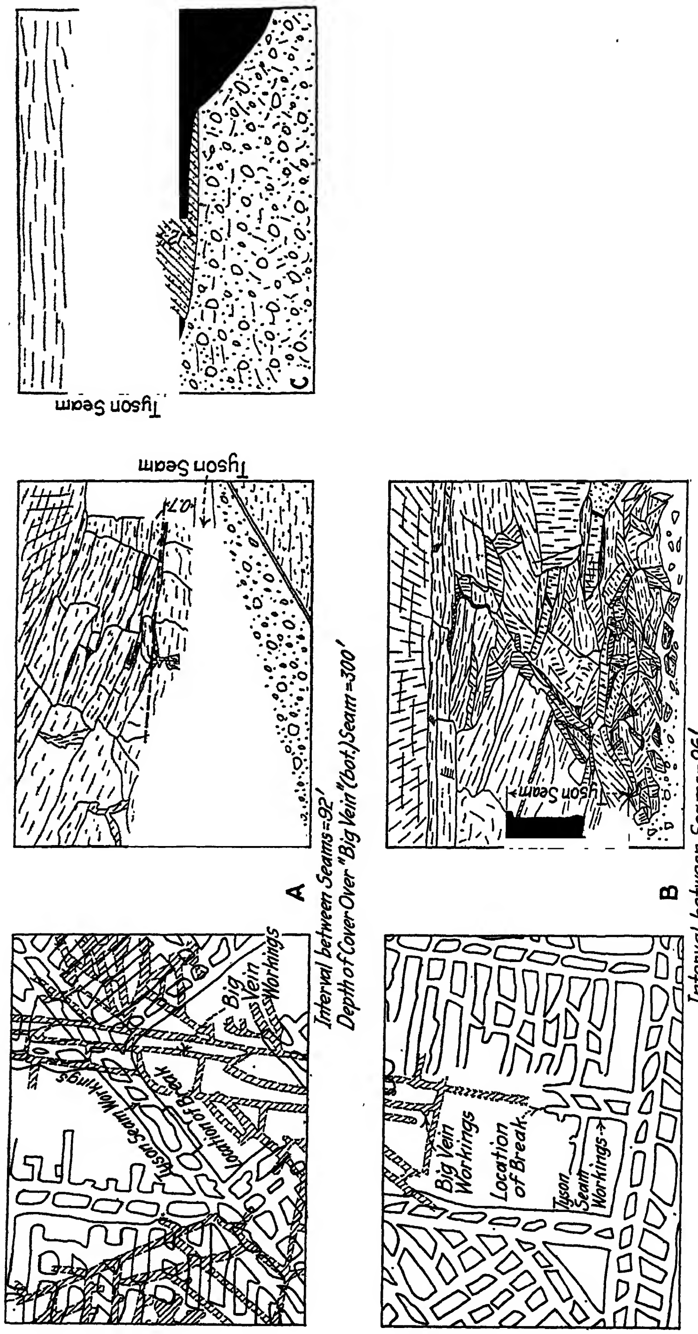


FIG. 19.—EXAMPLE OF SUBSIDENCE IN A MINE IN ALLEGHANY COUNTY, MD.

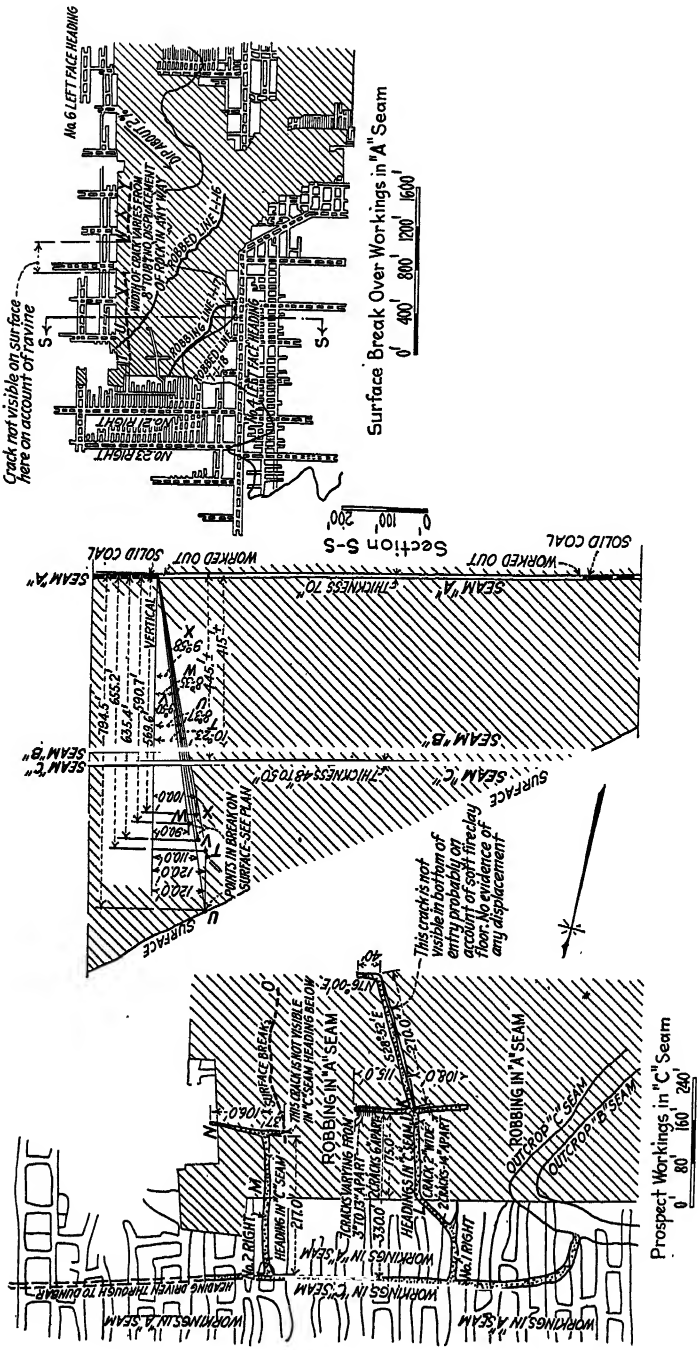

Surface Break Over Workings in "A" Seam
0' 400' 800' 1200' 1600'
No. 6 LEFT FACE HEADING
DIP ABOUT 2°
WIDTH OF CRACK VARIES FROM 8" TO 18" NO DISPLACEMENT OF ROCK IN ANY WAY
ROBBED LINE 1-1-16
ROBBING LINE 1-1-17
ROBBED LINE 1-1-18
No. 4 LEFT FACE HEADING
No. 21 RIGHT
No. 23 RIGHT
Crack not visible on surface here on account of ravine
Section S-S
0' 100' 200'
SEAM "A"
SEAM "B"
SEAM "C"
SOLID COAL
WORKED OUT
THICKNESS 70"
THICKNESS 48 TO 50"
VERTICAL
SURFACE
POINTS IN BREAK ON SURFACE-SEE PLAN
This crack is not visible in bottom of entry probably on account of soft fireclay floor. No evidence of any displacement
SURFACE BREAK
THIS CRACK IS NOT VISIBLE IN "C" SEAM HEADING BELOW
ROBBING IN "A" SEAM
HEADING IN "C" SEAM
HEADINGS IN "C" SEAM
No. 2 RIGHT
No. 1 RIGHT
WORKINGS IN "A" SEAM
WORKINGS IN "C" SEAM
HEADING DRIVEN THROUGH TO DUNBAR
OUTCROP "C" SEAM
OUTCROP "B" SEAM
Prospect Workings in "C" Seam
0' 80' 160' 240'

FIG. 20.

FIG. 20.—SURFACE BREAKS OVER A COAL MINE, SOUTHWESTERN VIRGINIA.

A close inspection of Nos. 1 and 2 right headings in C seam showed no displacement at points L, M and N or at any other point.

ELEVATION OF POINTS ON SURFACE BREAK, FEET

	T	U	V	W	X	Y
Surface	2593.2	2732.5	2576.4	2528.7	2507.6	
Bottom-A-seam	1938.0	1938.0	1941.0	1942.0	1938.0	

CO-ORDINATES OF POINTS

	T	U	V	W	X	Y
Horizontal	120.0	120.0	110.0	90.0	100.0	
Vertical	655.0	794.5	635.4	590.7	569.6	

GENERAL NOTE

Referring to note on workings in C seam. A careful examination of these workings over the worked out area in A seam failed to show any indication whatever that the A seam had been removed.

Station J in No. 1 right heading in C seam at a point 20 ft. outside of station J the right hand corner of heading cut into a crack about 2 in. wide which extended up into the roof and showed a separation of the coal seam the entire height K of seam. From this point to station K, the right rib of the heading follows this crack which thus shows on the map the general direction of the break. The bottom was soft in the heading and the crack could not be located in same though undoubtedly it was originally there. Upon the completion of above headings in C seam a very careful examination of the mining conditions was made and the following was recorded:

At station K heading driving to right, +4 ft. crack 2 in. wide (this is the original crack extending along right rib from station J to station K) + 4 ft. 11 in. crack parallel to first crack ½ in. wide (coal cracked down to pavement but no crack is found in bottom; no displacement is visible) + 32 ft. 8 in. and +33 ft. 10 in., 2 cracks about ½ in. wide; no displacement; roof and coal normal. Crack extends to bottom of seam +91 ft. 5 in. Faint crack opened up about a ¼ in. wide in roof +108 ft. Face of coal and roof normal in every respect. The roof over this coal changes from sandstone at K to 8 in. (approx.) of drawslate of face of place. With the exception the cracks the roof was absolutely normal and no trouble was encountered on account of the cracks.

Station K heading driven to left, +22-ft. crack in roof about ½ in. wide which extends in same direction across heading as the large crack found between J and K.

+81, and +81 ft. 6-in. cracks ¼ in. wide across roof and down in the coal. No crack in bottom is observed.

+103 ft., +103 ft. 9 in., +104 ft. 3 in., +105 ft. 4 in., +105 ft. 8 in., +106 ft. 2 in., + 106 ft. 5 in., location of seven fine cracks varying from ¼ in. to ½ in. wide which were observed for 3 ft. 5 in. commencing at 103 ft. The air current came down into the entry with sufficient force to be very noticeable from the fourth crack and was a great aid in ventilating the heading during driving.

+115 ft. face of entry. The roof from station K was sandstone the entire distance to the face. Roof and coal conditions were absolutely normal with exception of the cracks and it was impossible to observe any difference in the seam either from a mining or shooting standpoint.

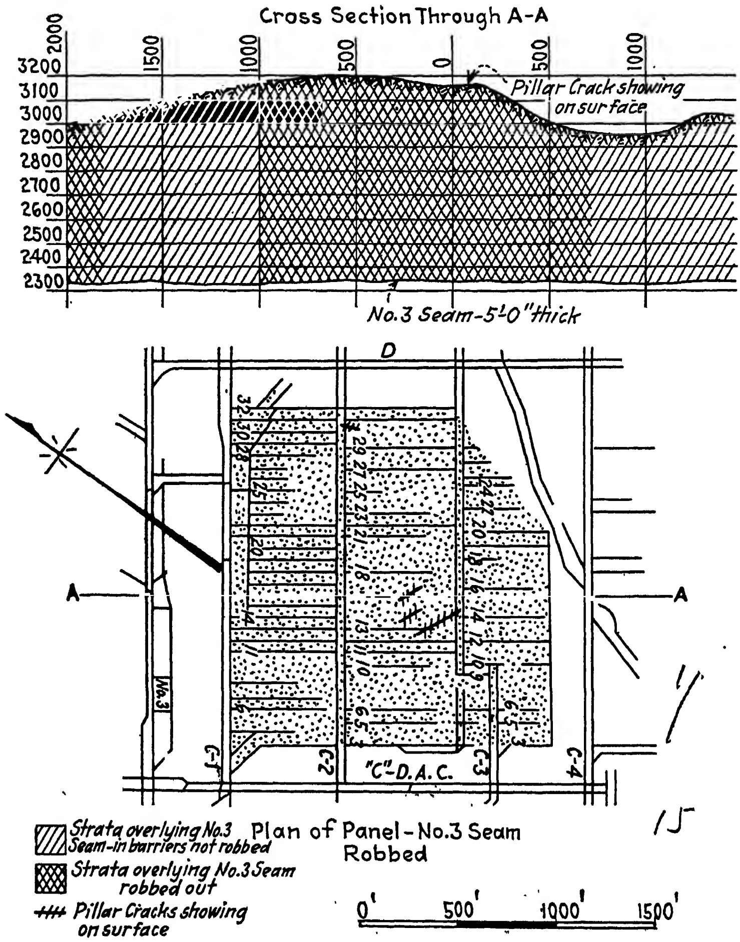


FIG. 21.—EXAMPLE OF SUBSIDENCE IN A MINE IN SOUTHERN WEST VA.

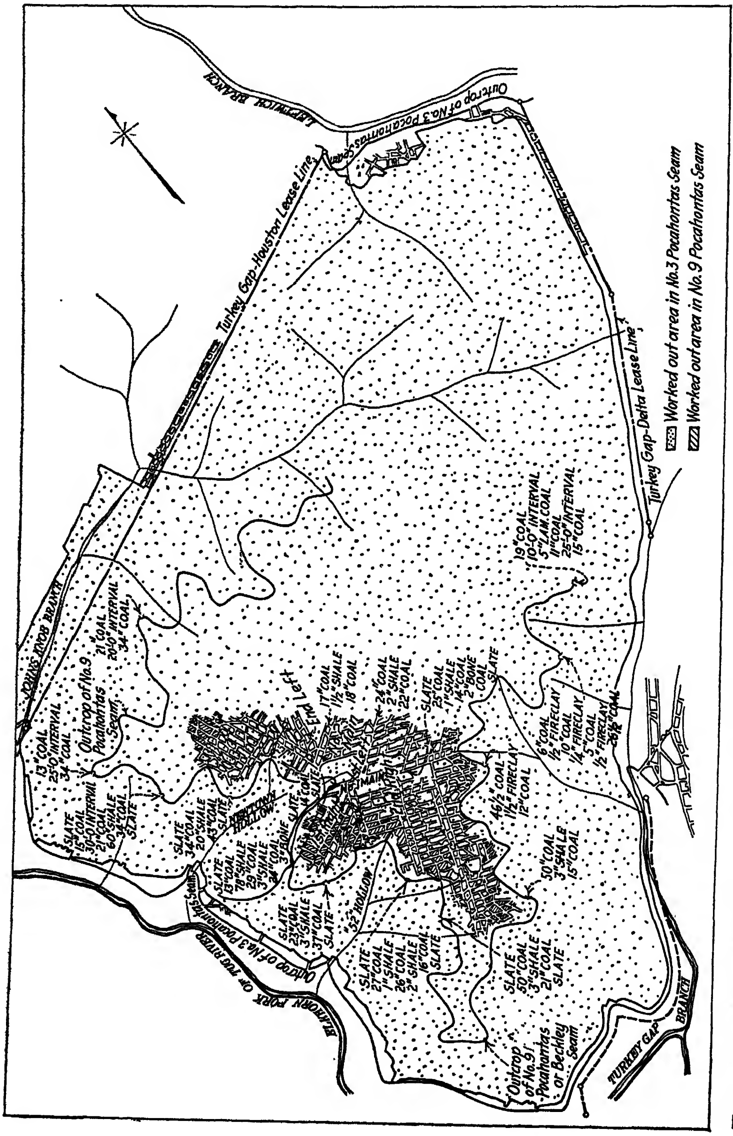


FIG. 22.—MAP OF WORKINGS NO. 9, POCAHONTAS SEAM OVER WORKED OUT AREA IN NO. 3 POCAHONTAS SEAM.

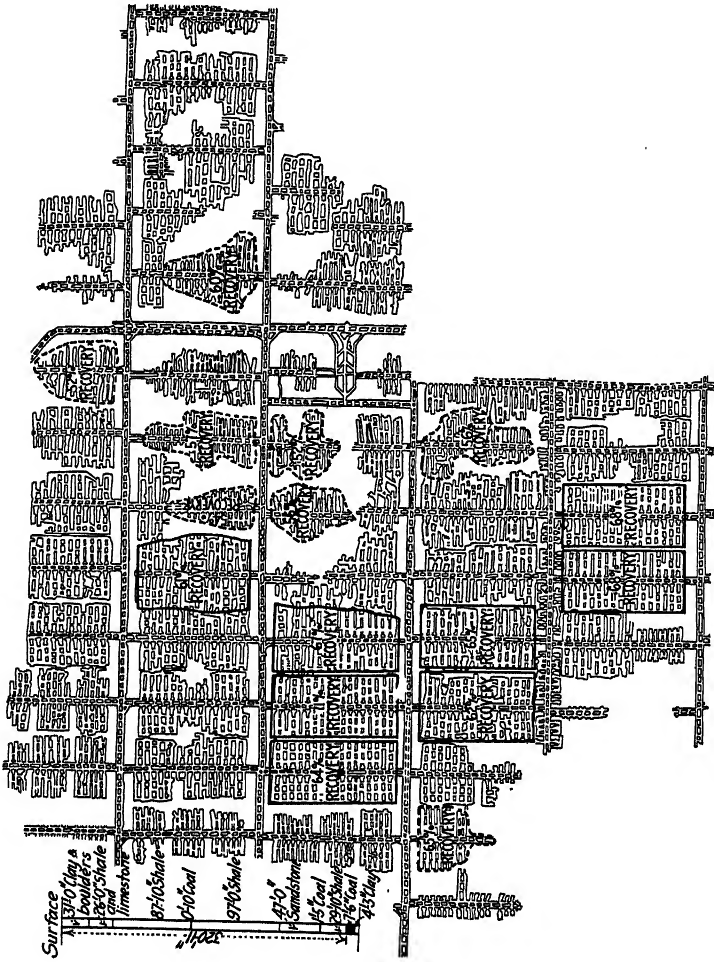


FIG. 23.—EXAMPLE OF SUBSIDENCE IN A COAL MINE, CHRISTIAN CO., ILL.

.... Section where squeeze occurred before coal was extracted due to soft wet bottom.

——— Section where coal was extracted before squeeze occurred.

Where fireclay bottom is reasonably hard and is not over 2 ft. thick, squeezes do not occur. Where fireclay is soft, contains water and the thickness exceeds five ft., squeezes occur regardless of the size of the pillars provided. Percentages of recovery shown are for areas inside lines only.

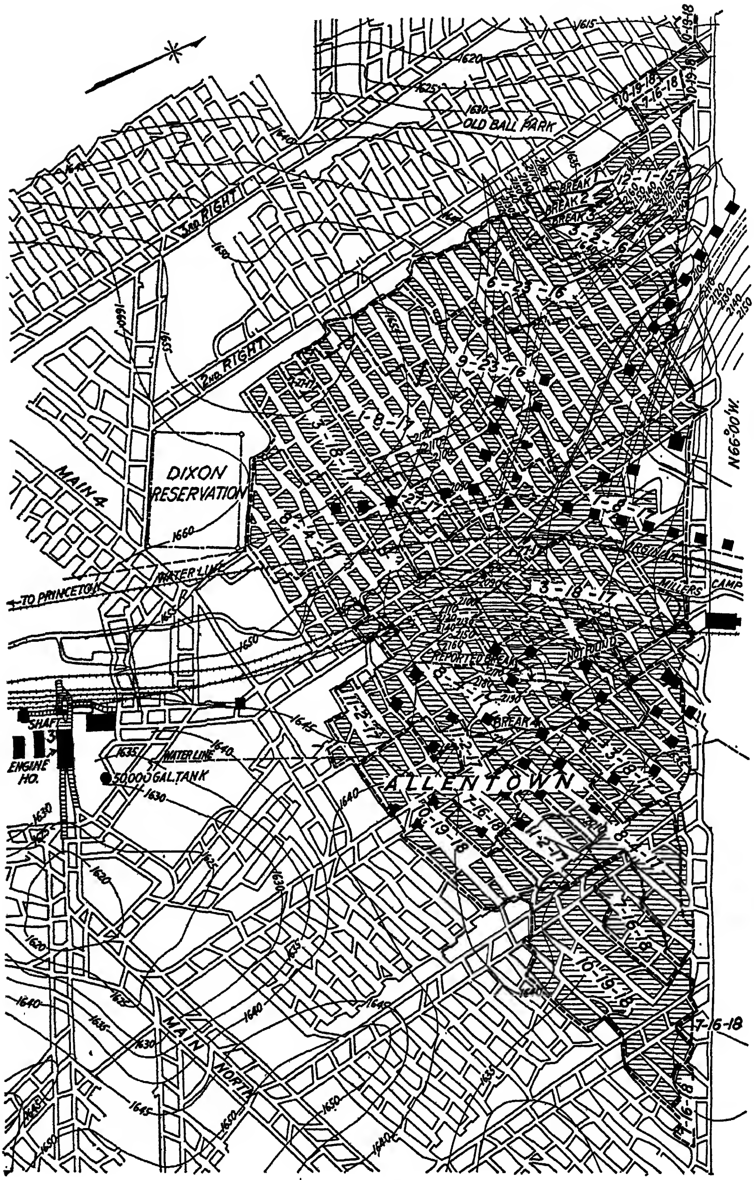
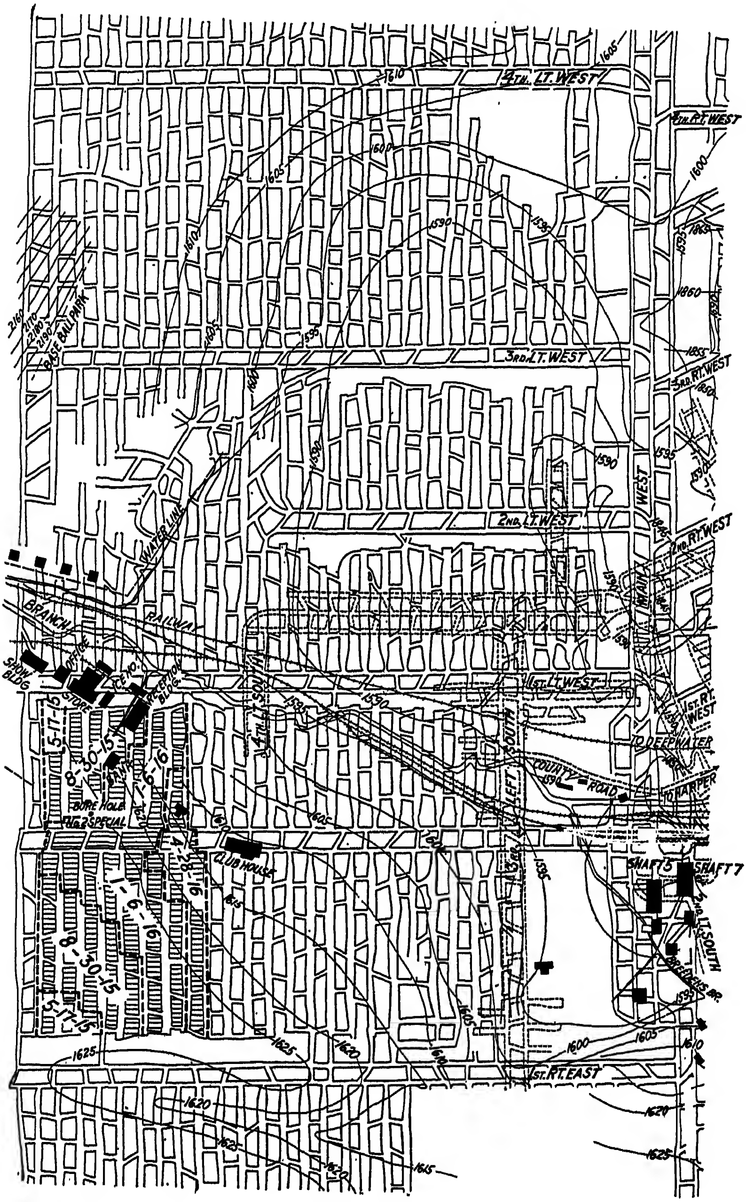


FIG. 24.—LOCATION OF SURFACE CRACKS AND



IMPROVEMENTS OVER BECKLEY SEAM WORKINGS.

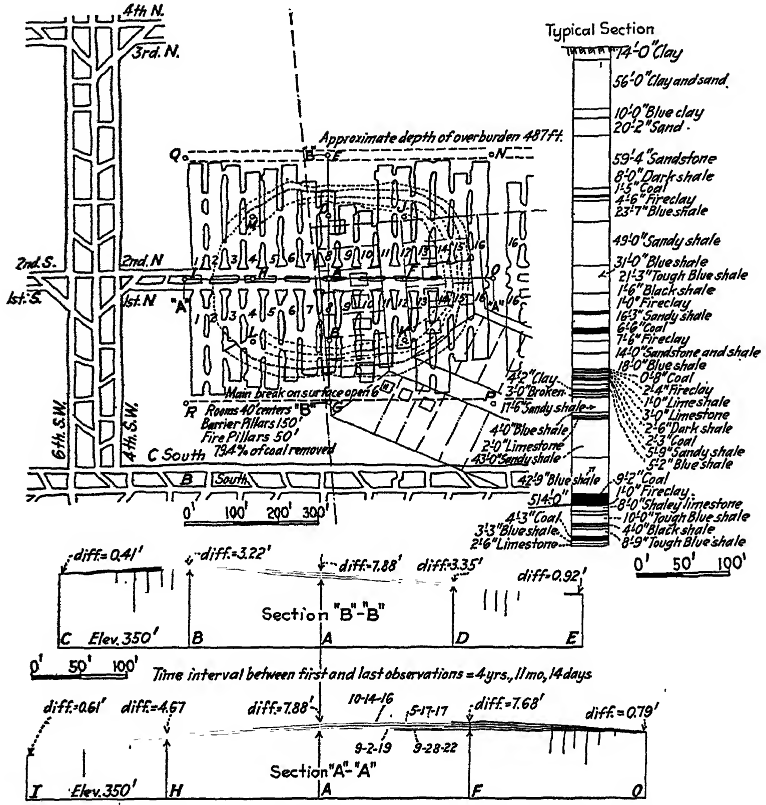


FIG. 25.—EXAMPLE OF SUBSIDENCE IN AN ILLINOIS COAL MINE. TIME INTERVAL BETWEEN FIRST AND LAST OBSERVATIONS, 4 YRS., 11 MOS., 14 DAYS.

ELEVATION OF MONUMENTS

Monument	10–14–16	5–17–17		9–12–19		9–28–22	
	Elev.	Diff.	Seam, Per Cent.	Diff.	Seam, Per Cent.	Diff.	Seam, Per Cent.
A	434.76	2.42	26.4	2.67	29.1	2.78	30.3
B	442.24	0.95	10.3	1.10	12.0	1.17	12.8
C	432.29	0.05	0.5	0.18	2.0	0.18	2.0
D	422.21	0.86	9.4	1.20	13.1	1.29	14.1
E	408.20	0.03	0.3	0.43	0.5	0.43	0.5
F	435.77	2.22	24.2	2.69	29.4	2.77	30.2
G							
H	417.58	0.76	8.3	1.89	20.6	1.99	21.4
I	393.91	0.07	0.8	0.30	3.0	0.30	3.0
J	424.96	1.48	16.2	1.85	20.2	1.91	20.8
K	448.56	0.62	6.8	0.75	8.2	0.81	8.8
L	415.63	0.41	4.5	0.66	7.2	0.71	7.8
M	409.81	0.32	3.5	0.99	10.8	1.08	11.8
N	400.19	0.02	0.2	0.34	3.7	0.34	3.7
O	424.22	0.03	0.3	0.38	4.1	0.38	4.1
P	442.83	0.06	0.6	0.11	1.0	0.11	1.0
Q	388.94	0.04	0.4	0.14	1.5	0.14	1.5
R	395.27	0.03	0.6	0.09	0.9	0.09	0.9

Note.—Approximate elevation of fire clay is −67.0 U. S. G. S.

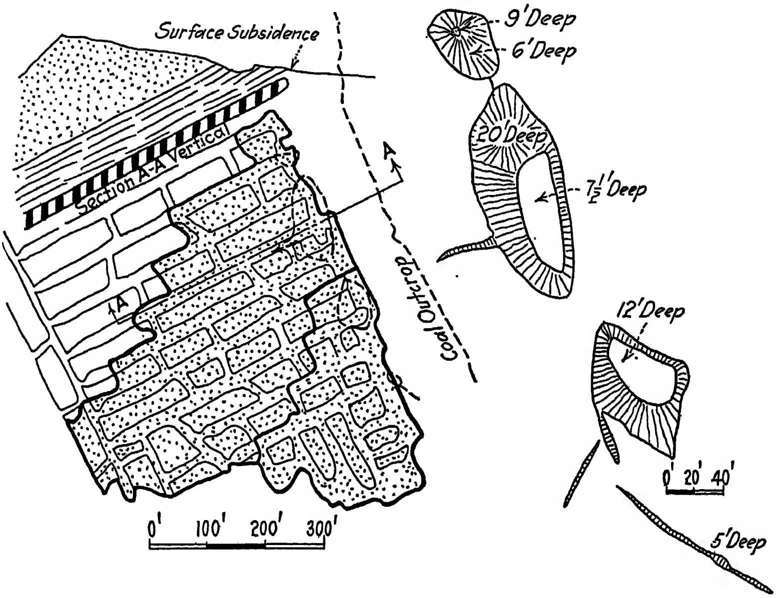


FIG. 26.—EXAMPLE OF SUBSIDENCE IN A UTAH COAL MINE.

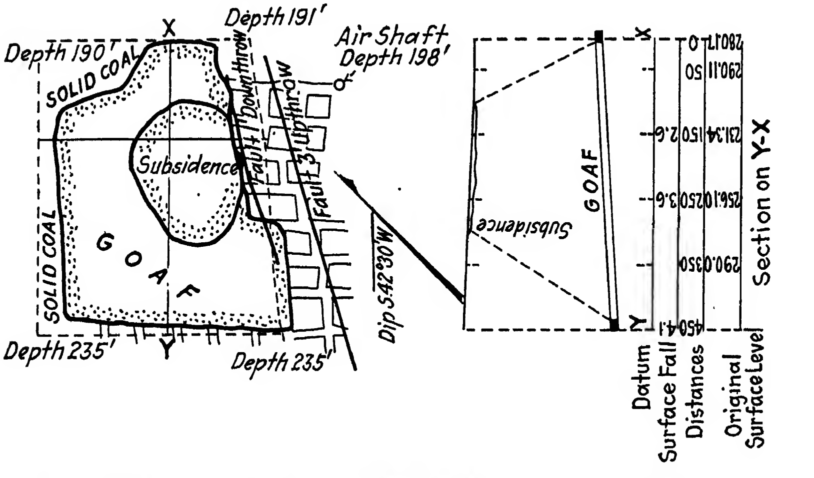


FIG. 27.—EXAMPLE OF SUBSIDENCE AT A COLLIERY IN INDIA. AVERAGE DEPTH, 210 FT. THICKNESS OF SEAM, 12 FT. DIP OF SEAM, 1 IN. 10 FT. AREA MINED BEFORE SUBSIDENCE OCCURRED, 120,400 SQ. FT., OR $8\frac{1}{2}$ BIGHAS; DIMENSIONS OF ABOVE, 450 BY 280 FT. MAXIMUM SUBSIDENCE OF SURFACE $3\frac{1}{2}$ FT.

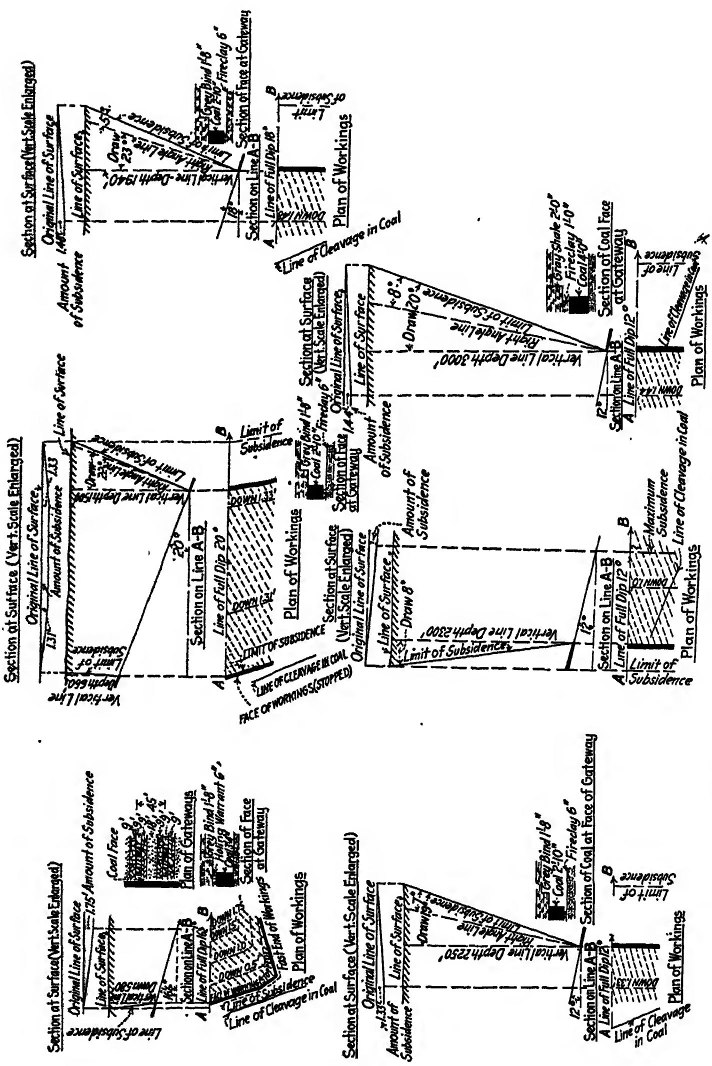
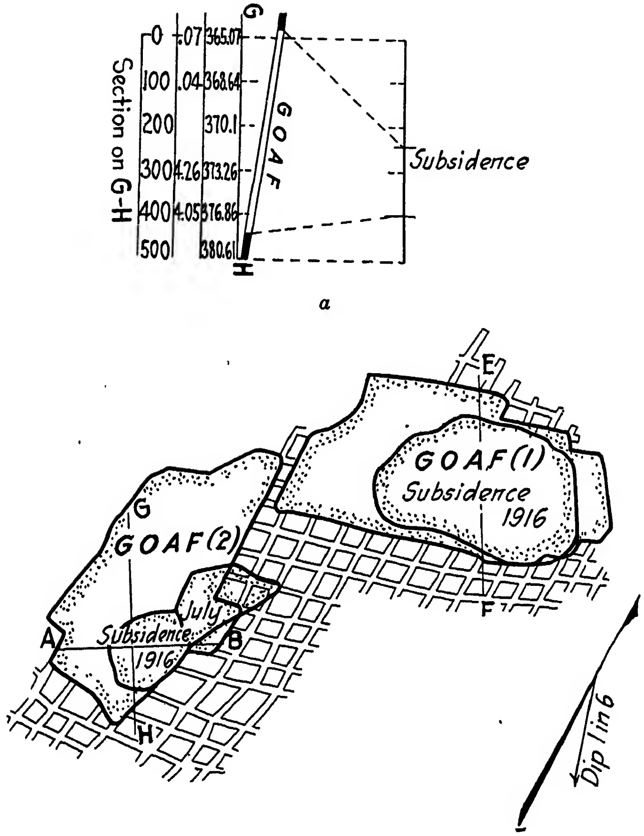
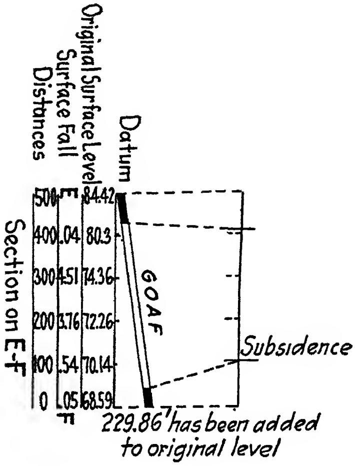


FIG. 28.—EXAMPLE OF SUBSIDENCE IN COAL MINES AT LANCASHIRE, ENGLAND.

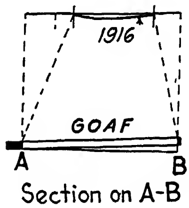

Section on G-H
G
H
0
100
200
300
400
500
.07
.04
4.26
4.05
365.07
368.64
370.1
373.26
376.86
380.61
GOAF
Subsidence
a
E
F
A
B
G
H
GOAF(1)
Subsidence
1916
GOAF(2)
July
Subsidence
1916
Dip 1 in 6

b

FIG. 29.



c



d

FIG. 29.—ANOTHER EXAMPLE OF SUBSIDENCE IN A COAL MINE IN INDIA. (*Continued.*)

	Goaf No. 1	Goaf No. 2
Average depth	240 ft.	300 ft.
Thickness of seam	15 ft.	15 ft.
Dip of seam	1 in 6 ft.	1 in 6 ft.
Area goaved before subsidence occurred	221,000 sq. ft. or 15 bighas	143,000 sq. ft. or 10 bighas
Dimensions of above	630 by 340 ft.	550 by 260 ft.
Area of subsidence	112,500 sq. ft.	24,300 sq. ft.
Maximum subsidence of surface	5 ft.	4 ft.
Note.—Maximum subsidence given in text as	5.1′ 6.5′	

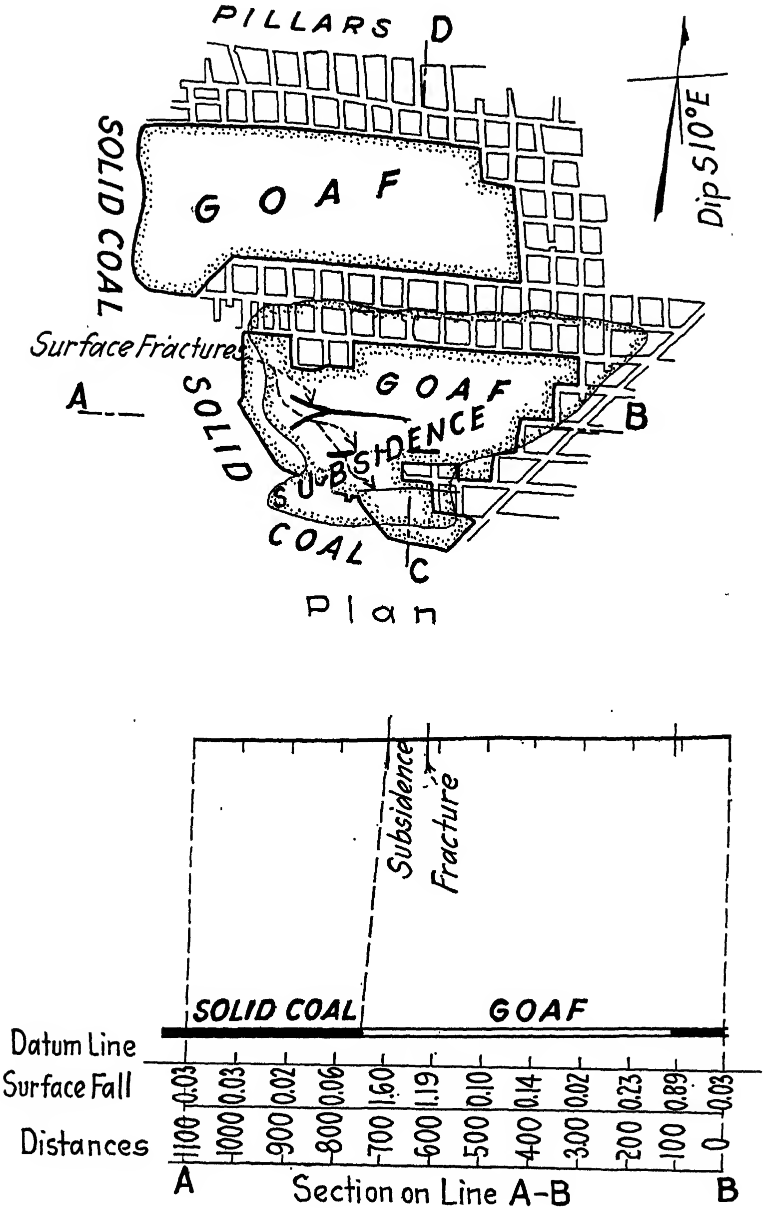

PILLARS
D
SOLID COAL
GOAF
Dip S10°E
Surface Fractures
A
SOLID COAL
GOAF
SUBSIDENCE
B
C
Plan
Subsidence
Fracture
SOLID COAL
GOAF
Datum Line
Surface Fall
0.03 0.03 0.02 0.06 1.60 1.19 0.10 0.14 0.02 0.23 0.89 0.03
Distances
1100 1000 900 800 700 600 500 400 300 200 100 0
A
Section on Line A-B
B

FIG. 30.

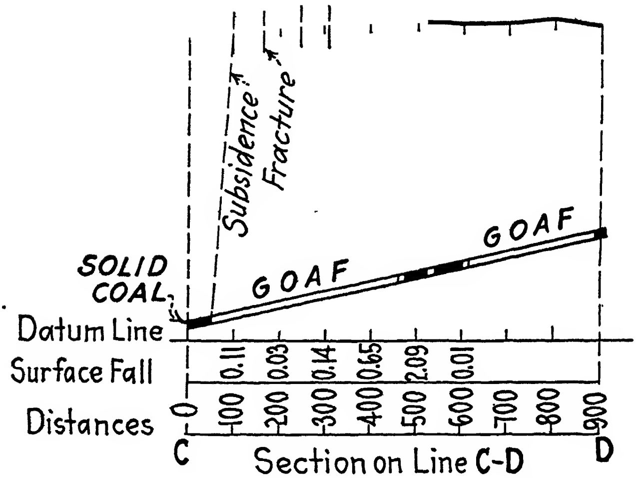


FIG. 30.—A THIRD EXAMPLE OF SUBSIDENCE IN A COAL MINE IN INDIA. AVERAGE DEPTH 590 FT.; THICKNESS OF SEAMS, 16 FT.; DIP OF SEAM, 1 IN. $5\frac{1}{2}$ FT.; MAXIMUM SUBSIDENCE, 2.09 FT. *(Continued.)*

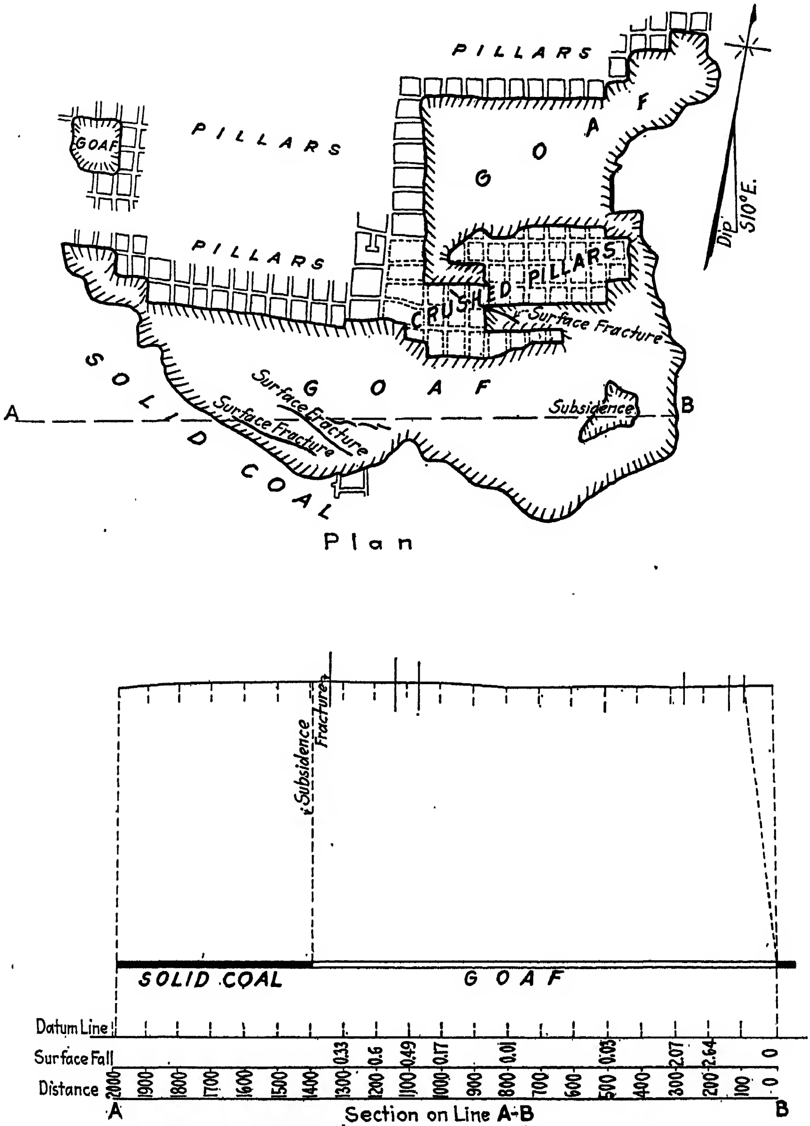


FIG. 31.—A FOURTH EXAMPLE OF SUBSIDENCE IN A COAL MINE IN INDIA. AVERAGE DEPTH 857 FT.; THICKNESS OF SEAM 15 FT.; DIP OF SEAM 1 IN. 5½ FT. AREA OF GOAF 23 ACRES. GOAFING COMMENCED IN 1914. SUBSIDENCE OF SURFACE IN 1917 WAS 2.1 FT., IN 1920, 2.84 FT. MAXIMUM SUBSIDENCE GIVEN IN TEXT IS 1.8.

Appendix A.—Form C, Coal Mining Subsidence

This information is for the use of the Committee on Ground Movement and Subsidence of the American Institute of Mining and Metallurgical Engineers. If for any reason you do not want the data you send, published, it will be treated as confidential.

1. Name of mine............................Location..................
2. Name of company.........................P. O. Address............
3. Kind of coal (anthracite, bituminous, lignite)..........................
4. Name of coal bed..
5. Thickness of bed..................Thickness of coal mined...........
6. If it has a thick parting, give thickness, position and character of parting.
   ....
7. Structure of coal (blocky, well marked faces and butts, friable) Does it spall off under pressure and under what conditions..............................
   ....
8. Ultimate crushing strength of coal, if tested, per sq. in......................
9. Kind of floor.........................Does it heave or squeeze.............
10. Kind of roof...........................strength......................
    Does it fall in rooms, and to what height..................................
11. Character of overlying rocks (give section if possible).......................
    Are they easily broken (state area usually completely removed before break occurs)..
12. Depth of coal bed below the surface where investigations made................
13. Is surface level or mountainous...
14. Dip of bed................If generally level is it very undulating...........
15. Is the mine wet.................Where does the water come from...........
16. Method of mining..................Room and pillar........................
    (a) Give width rooms and of pillars.......................................
    (b) If rooms are panels, give size of panels..............................
    (c) Are pillars extracted..................To what extent.................
    (d) Are panel and chain pillars left in...................................
    (e) What is method of extraction................Give sketch...............
    (f) What is the average thickness of gob material if it were spread out in the goaf..
    (g) Average height of goaf....................before roof breaks............
    (h) In extracting pillars how does roof break.............................
    (i) Does weight tend to ride over panel or chain pillars....................
    (j) Is squeezing up of the floor sometimes experienced......................
17. Method of mining, longwall
    (a) Type of longwall, advancing in circle, panel longwall, retreating longwall......
    (b) Is undercutting done in coal or in underclay............................
    (c) Is there draw slate, can it be held up..................................
    (d) Are the roadways "brushed"........Thickness of brushing................
    (e) Are pack walls built..................Width............................
    (f) To what extent is goaf filled by refuse from draw slate, underclay, partings, etc..
        Estimate average thickness of broken material put into goaf..............
    (g) Does the roof bend or break..
    (h) Angle or break in roof.............inclined toward goaf................

(*i*) Average lineal advance of face, per mouth............feet...............
(*j*) What is the ratio of rock refuse hoisted out of mine to coal hoisted, figured by cars or volume.

18. What is the character of subsidence, caveins, holes, etc. (Give specific measurements wherever possible)..
19. How soon after pillars are drawn was subsidence or cracking noticed..........
....
20. Has subsidence stopped, or is it still continuing at time of making report.......
21. Amount of subsidence at certain monuments, towns, etc. on specified dates (if possible, send print showing relation of movements and areas from which pillars have been removed)..
....
22. Photographs of damage to surface improvements.
23. Distance or angle of break or subsidence from line of pillar removal; is break or subsidence over solid coal or over withdrawn areas........................
24. In case of longwall, note character of cracks in surface improvements and whether there is an opening of cracks before subsidence begins.....................
25. Effect of pillars left in mine on character of subsidence......................
26. Effect of dip of bed on angle of "break" or "draw"........................
27. Effect of slope of surface in hilly or mountainous country on angle of "break" or "draw"..Submit profile.
28. Effect of known overlying rocks and earth overburden on angle of "break" or "area"
29. It is urged that lines of survey monuments of concrete or pipe with the bottom set below frost line, be established by the company in typical localities, crossing over panels, or longwall faces, and that at regular intervals surveys be made to determine the fact and extent of subsidence with relation to the position of the line of pillar drawing or of the longwall.
30. If mine is subject to "bumps" give particulars of occurrence, a map showing localities and geologic cross-sections.
31. Signature of observer...

Appendix B. Bibliography on Surface Subsidence Due to Mining Operations

1913

A. H. Goldreich: "Die Theorie der Bodensenkungen in Kohlengbieten." Julius Spregner, Berlin, (1913) 260 pp. Discusses the theory of subsidences in coal regions. Gives theories of subsidences of ground and railroads as a consequence of coal mining. Discusses the influence of ground subsidence due to coal mining on the condition of railroads.

1916

"More Mine Cave-Ins Threaten Parts of Scranton." *Eng. News*, (1916) **76**, 280. Discusses imminent danger to surface property and means of avoiding. Photographs show damage to streets and buildings.

"Some British Subsidences." *Coal Age*, (1916) **10**, 642. Explains the situation in Northwich, England, and shows in pictures damage done to surface property.

L. E. Young: "Surface Subsidence in Illinois Resulting from Coal Mining." Ill. State Geol. Survey Cooperative Coal Mining Ser., *Bull.* 17 (1916). 100 pp. Describes preliminary survey and investigations.

L. E. Young and H. H. Stock: "Subsidence Resulting from Mining." Univ. Ill. Eng. Exp. Station, *Bull.* 91, (1196) 179 pp.; abstr in *Coal Age*, **11**, 238. Con-

tains seven chapters covering the nature and extent of the subsidence problem, the geological conditions affecting subsidence, the theories and general principles of subsidence, engineering data and observations, laboratory experiments and data, protection of objects on the surface and legal considerations. Pp. 180 to 205 give an extensive bibliography of American and foreign literature on subsidence in mining.

1917

"Alaska Treadwell Mines Flooded." *Eng. & Min. Jnl.* (1917) **103,** 761. Tells how the shore area subsided and the mines were flooded with salt water in spite of bulkheads and other preventive measures.

J. F. Brown: "Some Theories on Mine Subsidence." *Coal Age,* (1917) **11,** 950 Analyzes the way in which the mine roof acts after part of the support is withdrawn. Stresses the effects of flux in the softer measures and points out how the plastic elements in roof, ribs and floor modify the action of the more elastic elements.

1918

"Anthracite Mine-Cave Situation." *Coal Age,* (1918) **14,** 598–601. Discusses work of mine-cave committees in making adjustments for damage done in Scranton, Pa.

A. Montgomery: "Earth Movements and Underground Conditions in Mines." *Monthly Jnl.,* Chamber of Mines of Western Australia, (1918) **17,** 119–27. Gives a detailed account of a fall of ground in the Great Boulder Mine, Kalgoorlie, which may have been due both to earthquake shocks and mining operations.

1919

Brychan: "Subsidence in Mine Working." *Sci. & Art. of Min.* (1919) **29,** 375, 470 Discusses theory of dome in relation to subsidence. Gives examples of success in completely removing coal from beneath a building.

R. W. Mayer: "Method Employed in Working the Crescent Mine." *Coal Age* (1919) **15,** 1028. Explains how squeezes are prevented by careful use of both machine and pick mining.

Vincenz, Pollack: "Uber Bodensenkungen durch Berg und Tunnelbau mit besonderer Berucksichtigung der Vorkomminisse und Versuche in Frankreich." *Ztsch.* des-Osterr. Ing. & Arch. Ver., (1919) **71,** 255, 287, 263, 321, 425. Discusses surface subsidence through mining and tunneling especially in reference to occurrences and experiments in France. Describes the artificial experiments made by Fayol.

W. D. Lloyd: "The Effect of Coal Mining on the Overlying Rocks and on the Surface." *Trans.,* **57,** 74–100. Calculates the amount of subsidence in coal mines. Discusses necessary allowances to be made, the amplitude of subsidence, the influence of faults and character of the overlying rocks on subsidence, the time factor in studying subsidence, and amount of settlement at various dates, shown in a table on p. 91.

1920

M. Delbronck: "Les affaisements du sol produits par les exploitations minieres," *Rev. Univ.* des *Mines* da la *Met.,* (1920) **6,** 49–54. Surface subsidence due to mining. Discusses theories on the question and rules for determining the zones affected.

H. Louis: "Compensation for Subsidences." *Trans.* (1920) **59,** 292–310; discussion, (1926), **60,** 240. Discusses the advisability of issuing a bill to cover subsidence compensation for all classes of mines. Disapproves such a bill, showing by

examples that no two types of mines could be governed by the same form of compensation.

H. G. Moulton: "Earth and Rock Pressures." *Trans.* (1920) **63**, 327. Describes subsidence over copper mining operations in Arizona and Nevada, and compares the angles of draw produced thereby with the corresponding angles observed in connection with subsidence over subway tunnels in New York City. Illustrated. Also discusses slopes in open cuts, and considers the mechanics of earth viewed as a weak solid.

R. E. Palmer: "Some Observations on Mining by the Open Cast or Stripping Method." *Trans.* (1920) **30**, 128–75. Pages 130, 138, 141, 143, 146, 149, 165–7 and 169 discuss the subject of subsidence in connection with mining operations.

Vincenz Pollack: "Uber Stutzpfeiler von Bauten in Senkungsgebieten." *Mont. Rundschau,* (1920) **12**, 123, 147, 163, 182, 200. Discusses supports for structures in regions affected by subsidence.

1921

R. C. Morgan: "Causes of Subsidences and the Best Safeguards for Their Prevention." *Proc.* South Wales Inst. Eng. (1921) **37**, 49–72; abstr. in *Coll. Guard.,* (1921) **121**, 795. Discusses subsidence and gives a table showing maximum movement, time taken, width of motive zone, average rate of subsidence per day, relative stress, rate of advance per year, and thickness of roof.

G. C. H. Whitelock: "Subsidence Due to Coal Mining." *Coll. Guard.* (1921), **121**, 109–10. Paper before Inst. of Mine Surveyors of Great Britain, Dec. 17, 1920. Discusses subsidence subsequent to mining at from 100 to 500 yds. below the surface.

1922

H. Louis: "Subsidence Theory." *Coll. Guard.,* (1922) **124**, 1215; also *Iron & Coal Tr. Rev.,* (1922) **105**, 717. (Abstr. from paper before Inst. of Min. Eng.). A commission acting under the Railway Clauses Act, concludes that a safe angle of draw is 25° 35′ applying to inclined as well as horizontal strata. Illustrated.

Nierhoff: "Der Einfluss von Bodensenkungen in Bergbaugebieten auf die baulichen Anlagen und den Betrieb der Eisenbahnen." [Subsidences in Mining Districts and Their Effect on Railway Buildings and Operation.] Archiv f. Eisenbahnwesen, (1922) **45**, 1165–1215. Gives causes for subsidence, with legal cases, assessment for damages and details of effect on foundations of buildings, walls, bridges and railroads.

"Subsidences Report of Mining and Geological Institute of India." *Trans.* Min. & Geol. Inst., India, (April, 1922) **16**, 145–66. Discusses observations made in the Bengal and Bihar and Orissa coal fields over a long period to determine the effects of the extraction of coal at various depths. The Committee concludes that subsidence is less where no packing is done and where pillars are completely removed. In no case has the shift of subsidence been great enough to take surface effect from the surface vertically above the excavated area.

1923

G. F. Burch: "Coal Mine Subsidence Damages Concrete Girder Bridge." *Eng. News* (1923) 90, 1046. Discusses partial failure of Spring Creek Bridge on the Beardstown Road, near Springfield, Ill.; the banks showed cracks, due to coal mine subsidence. Photographs show damage done.

E. A. Gowers: "Mining Subsidence." *Iron & Coal Tr. Rev.* (1923) 107, 662–3, and 666–7. (Abstr. from evidence before Royal Commission on Mining Subsidence.) Discusses subsidence according to surface movements and underground movements as each pertains to the law. Proposes remedies.

H. LOUIS: "A Contribution to the Theory of Subsidences." *Trans.* (1923) **64**, 257–73. Discusses the importance of giving more consideration to surface subsidence. Gives diagrams illustrating Coulombs theory and mathematical calculations for safe angle of draw. Due to lack of authentic information and actual examples the author does not give a definite calculation for vertical subsidence. Discussion follows.

G. S. RICE: "Some Problems in Ground Movement and Subsidence." *Trans.* (1923) **69**, 374–93; abstr. in *Min. & Met.* (1923) **4**, 480–81. Shows that although empirical formulas are desirable, each locality is an individual problem and that the subject of subsidence must be treated according to local conditions. Declares that the room and pillar system does not cause subsidence unless pillars are too thin; that surface subsidence is inevitable after a lapse of time when pillars are wholly or partly removed, and that longwall systems will always cause surface subsidence.

J. J. RUTLEDGE: "Subsidence in Two Oklahoma Coal Mines." *Min. & Met.*, (1923) **4**, 481–2. Tells the story of the cave in a Union Coal Co. mine at Adamson, Okla. Pillar-and-room mining used. Also gives some facts on the subsidence at the Crow Coal Co. mine at Henryetta. Photograph shows cracks in the earth.

I. C. F. STATHAM: "Subsidence and Shaft Pillars." *Coll. Guard.* (1923) 125, 325, 387 and 449. Describes the parts played by gravity and by contraction in bringing about subsidence; diagrams the action of both these forces, and gives development of curves in rigid and in flexible strata. Illustrated.

J. WEISSNER: "Subsidence and Shaft Pillars." *Coll. Guard.* (1923) **126**, 335. States, contrary to a popular opinion, that in propagation of fracture from coal face upward, the slipping of lateral boundaries proceeds dependent upon the natural angle of repose of the rocks. Gives comparative data on dimensions of pillars.

W. O. WOOD: "Subsidence in Coal Districts." Trans. (1923) **65**, 178–86 (Not available).

1924

C. H. BAILEY: "Mining Subsidence." *Coll. Guard.* (1924) **127**, 1183. Distinguishes between subsidence and damage due to subsidence. Gives calculations of subsidence, with a discussion on causes, the legal position and the standpoint of the surveyor, and discussions of several engineers.

"Hydraulic Stowage at Home and Abroad." *Coal Age*, (1924) **25**, 355. Discusses two papers on subsidence delivered before the Amer. Inst. Min. Eng. Concludes that a certain amount of surface subsidence is unavoidable, therefore, control is a more important study than prevention.

"Subsidence: How Far It Extends, Its Depth and Rapidity of Movement." *Coal Age*, (1924) **26**, 557 (abstr. from T. A. O'Donahue's testimony before Royal Commission on Mining Subsidence). Subsidence is said to stretch furthest beyond the coal face when working advance to the dip. The sag is less pronounced where the seam is deepest.

APPENDIX C.—*Data of Subsidence in Foreign Mines*

No.	Location	Seam Worked	Thickness		Depth of Cover	Subsidence				
			Total	Mined		Due to	Amount and Character	Thickness of Seam, Per Cent.	Begun after Robbing	Ended
1	South Kirby Colliery, Wales		3 ft. 9 in.	4 ft. 9 in.	2108 ft.	Pillar withdrawal	Surface lowered 3.47 ft.; shaft pillar in an upper bed crushed.	73	4 yr.	11 yr. 8 mo.
2	Derbyshire		5 ft.	5 ft.	1595 ft.	Pillar withdrawal	Maximum 1.74 ft.	35		
3	South Wales		3 ft. 4 in.	3 ft. 4 in.	360 ft.	Pillar withdrawal	Workings in upper seam affected.			
4	Bent Colliery, Lanarkshire		5 ft. 6 in.	5 ft. 6 in.	650 ft.	Pillar withdrawal	Maximum 4.0 ft.; average 3.76 ft.	73	4 mo.	4 yr.
5	Hickletŏn Main Colliery					Pillar withdrawal	Amount 4.5 ft.			
6	Teversal & Peasley Collieries	Top hard	5 ft. 6 in.	5 ft. 6 in.	600 ft.	Pillar withdrawal	Walls of church cracked and were rebuilt; max. crack 8 in.			
7	Stuffynwood Hall	Top hard	5 ft.	5 ft.	1500 ft. to 1700 ft.	Pillar withdrawal	Maximum 1.74 ft.; minimum 1.34 ft., under a building.	30	4 mos.	2 yr. 8 mo.
8	Downhill Reservoir	Mandlin	5 ft. 6 in. to 6 in.	5 ft. 6 in. to 6 in.	1,770 ft.	Pillar withdrawal	2 in. to 3¾ in., large cracks.		3 yr.	
9	Whettleford Pump Station	Slate 2 yd.			850 ft.	Pillar withdrawal	9.0 ft. Slate seam was packed for 9 acres. Top seam was not packed.		14 mo.	12 yr.
10	Robinson's End Reservoir	Two yard	5 ft. 6 in.	5 ft. 6 in.	915 ft.	Pillar withdrawal	3.71 ft. Ended 4 yrs. after working.	67	3 mo.	21 yr.
10-A	Robinson's Filters	Two yard	5 ft. 6 in.	5 ft. 6 in.	915 ft.	Pillar withdrawal	4.45 ft.	80		11 yr.
11	Castleford, Yorkshire		4 ft. to 4 ft. 9 in.	4 ft. to 4 ft. 9 in.	1650 ft.	Pillar withdrawal	Aver. 2.25 ft.	56	4 mo.	4 yr.
12	India, Case No. 2		15 ft.	12 ft. (Text does not agree with drawing)	240 ft. 300 ft.	Pillar withdrawal	Max. 5.1 ft. Max. 6.5 ft.	42 54		
13	India, Case No. 9	Ghusick	12 ft.	12 ft.	210 ft.	Pillar withdrawal	Max. 3.5 ft.	28		
14	India, Case No. 16	·	16 ft.	16 ft.	590 ft.	Pillar withdrawal	Max. 2.1 ft.	13		
15	India, Case No. 15		15 ft.	15 ft.	857 ft.	Pillar withdrawal	Max. 1.8 ft.—surface cracked.	12	2 yr.	
16	Ayrshire		(Text does not agree with drawing)	6 ft.						

17			8 ft.		321 ft.		2.08 ft.	25		
18	South Yorkshire..................		8 ft.		400 ft. to 700 ft.	Pillar withdrawal	1.92 ft.	23		
19	Doncaster......................				1800 ft.	Pillar withdrawal	1 ft. 6 in. to 1 ft. 9 in.	40 to 75	5 yr.	
20	South Yorkshire..................				1200 ft. to 2100 ft.	Pillar withdrawal				
21					1920 ft.	Pillar withdrawal	Max. 2.75 ft. in 67 days.	48		
22					1650 ft.	Pillar withdrawal	Max. 2.83 ft. in 92 days.	43		
23					2140 ft.	Pillar withdrawal	Max. 3.41 ft. in 122 days.	57		
24					2410 ft.	Pillar withdrawal	Max. 3.16 ft. in 126 days.	56		
25			3 ft.	3 ft.	1050 ft.	Pillar withdrawal	Damaged a house.			
26			5 ft.	5 ft.	360 ft.	Pillar withdrawal	Average fall subsidence 3.5 ft.	70	Closely followed mining 6 mo.	3½ yr.
27			4 ft.	4 ft.	990 ft.	Pillar withdrawal	Average fall subsidence 2.6 ft.	64		4 yr.
28	Nottinghamshire-Newstead Abbey..		3 ft.	3 ft. 8 in.	1680 ft.	Pillar withdrawal	Max. 1.9 ft.	51	9 mo. before coal was exc.	7 yr.
29	Nottinghamshire.................		3 ft.	3 ft. 8 in.	1680 ft.	Pillar withdrawal	Building rose 4 in. at one point.			
30	West of England.................		7 ft. 6 in.	7 ft. 6 in.	216 ft.	Pillar withdrawal	Railroad arch span 20 ft. not affected.			
31	West of England.................		5 ft. 0 in.	5 ft. 0 in.	600 ft. to 900 ft.	Pillar withdrawal	Railroad tunnel only slightly cracked; no traffic delays.			
32	South Staffordshire...............		8 ft. 3 in.		1200 ft.	Pillar withdrawal	Max. 4 ft. in reservoir.	48		
33	South Staffordshire...............		30 ft.		432 ft.	Pillar withdrawal	Max. 13.3 ft. under a canal.	44		
34	Castleford Sewage Works..........		4 ft. to 4.5 ft.		603 ft.	Pillar withdrawal	Average fall subsidence 3.13 ft.; surface not broken.	78		
35	England.........................		3 ft. 9 in.	3 ft. 9 in.	324 ft.	Pillar withdrawal	1.84 ft. under houses.	49		
36	England.........................		4 ft. 0 in.	4 ft. 0 in.	288 ft.	Pillar withdrawal	Building fractured; over 150 ft. solid block around building.			
37	England.........................		4 ft. 0 in.	4 ft. 0 in.	306 ft.	Pillar withdrawal	Fracture line 10 ft. from bldg.; amount left around it 130 ft.			
38	Yorkshire.......................		3 ft. 10 in.	3 ft. 10 in.	1066 ft.	Pillar withdrawal	Max. 1.75 ft.		9 mo.	4 yr. 9 mo.
39	Scotland........................		2 ft. 9 in.		500 ft.	Pillar withdrawal	House subsided 2.16 ft.	78		2 yr. 4 mo.

APPENDIX C.—*Data of Subsidence in Foreign Mines*—(*Continued*)

No.	Location	Seam Worked	Thickness		Depth of Cover	Subsidence				
			Total	Mined		Due to	Amount and Character	Thickness of Seam, Per Cent.	Begun after Robbing	Ended
40					240 ft.	Pillar withdrawal	Small; open crack.			
41					600 ft.	Pillar withdrawal	Small			
42	Nuneaton		17 ft. 6 in.	All, less waste	1020 ft.	Pillar withdrawal	Walls cracked.			
43	Natal, South Africa	Coal Sandstone Coal	4 ft. 0 in. 4 ft. 9 in. 5 ft. 6 in.	9 ft. 6 in.	250 ft.	Pillar withdrawal	Max. 4 ft. 0 in.	42		
44	Lancashire, England	Upper Seam	3 ft. 4 in.	4 ft. 0 in.	580 ft. to 830 ft.	Mining	Max. 1.75 ft.; packing mat'l. av. 10 in.			
45	Lancashire, England	Upper Seam	2 ft. 10 in.	5 ft. 0 in.	660 ft. to 1500 ft.	Mining	1.31 ft. to 1.33 ft.		2 yr.	12 yr.
46	Lancashire, England	Upper Seam	2 ft. 10 in.	5 ft. 0 in.	1740 ft.	Mining	1.48. Packing mat'l. av. 10 in.			
47	Lancashire, England	Lower Seam	2 ft. 10 in.	5 ft. 0 in.	2140 ft.	Mining	1.33 ft.			
48	Lancashire, England	Lower Seam	2 ft. 10 in.	5 ft. 0 in.	2570 ft.	Mining				
49	Lancashire, England	Lower Seam	4 ft. 0 in.	7 ft. 0 in.	2970 ft.	Mining	1.44 ft. Packing mat'l. av. 1.5 in.	36	2½ yr.	12 yr.

APPENDIX C.—*Data of Subsidence in Foreign Mines*—(*Continued*)

No.	Location	Character of Strata over Coal	Draw			Dip of Strata	Mining Methods		Coal Removed, Per Cent.
			Distance Ahead	Angle Forward	Angle Back		Kind	Spacing, Etc.	
1	South Kirby Colliery, Wales	1 ft. of clod above coal, Beamshaw seam 3 ft. at 1600 ft. & Barnsley 9 ft. at 1800 ft. had been worked.				5.6%			
2	Derbyshire						Longwall		
3	South Wales		150 ft.				Longwall		
4	Bent Colliery, Lanarkshire		1 in 7.4 ft.	Av. 8°		5%	Longwall	Openings well stowed.	
5	Hickleton Main Colliery		433 ft.				Longwall		100
6	Teversal & Peasley Collieries	Mainly rock (limestone); no faults.	190 ft.	16°			Longwall		100
7	Stuffynwood Hall	Limestone and shale 78 ft.; coal measures 1553.	240 ft. to 315 ft.	$12\frac{1}{2}$°		4.2%	Longwall		
8	Downhill Reservoir	Limestone 320; coal measures 1450.					Longwall		None
9	Whettleford Pump Station	140 ft. sandstones, balance marl.				7.2%	Longwall		100
10	Robinson's End Reservoir	140 ft. sandstones, balance marl.				7.2%	Longwall	2.85 acres under, packed solid.	100
10-A	Robinson's Filters	140 ft. sandstones, balance marl.				7.2%	Longwall	2.85 acres under, packed solid.	100
11	Castleford, Yorkshire						Longwall		
12	India, Case No. 2	Mainly heavy, strong sandstone.	(Text does not agree with drawing)		4 to 43°	16.7%	Room and pillar	Rooms 90 ft. centers, about 15 ft. wide.	Probably 80
13	India, Case No. 9	Mainly heavy, strong sandstone.			26 to 33°	10%	Room and pillar	Rooms 48 ft. centers, about 12 ft. wide.	Probably 80
14	India, Case No. 16	Mainly heavy, strong sandstone.			0 to 4°	18%	Room and pillar	Rooms 48 ft. centers, about 12 ft. wide.	Probably 80
15	India, Case No. 15	Mainly heavy, strong sandstone.	Text does not agree with drawing)		0 to 6°	18%	Room and pillar	Rooms 48 ft. centers, about 12 ft. wide.	Probably 80
16	Ayrshire			15°			Longwall		
17							Longwall		
18	South Yorkshire			5.7 to 9.5°					
19	Doncaster	50 ft. sand just under soil; balance rock.					Longwall		
20	South Yorkshire		1/6 to 1/10 depth						

APPENDIX C.—*Data of Subsidence in Foreign Mines*—(*Continued*)

No.	Location	Character of Strata over Coal	Draw			Dip of Strata	Mining Methods		Coal Removed, Per Cent.
			Distance Ahead	Angle Forward	Angle Back		Kind	Spacing, Etc.	
21									
22						8.3%	Longwall		
23						8.3%	Longwall		
24						8.3%	Longwall		
25						8.3%	Longwall		
26		Binds and shales; no massive rock bed.	600 ft.			Nearly level			
27		Binds and shales; no massive rock bed.				Nearly level			
28	Nottinghamshire-Newstead Abbey		270 ft.				Longwall	All coal removed.	100
29	Nottinghamshire		200 ft.				Longwall	Pillar 408 ft. diameter left under house.	None
30	West of England						Longwall	All coal removed.	
31	West of England						Longwall	All coal removed.	
32	South Staffordshire	Another seam 6 ft. 3 in. thick, 66 ft. below, was also worked to within 50 yds. of reservoir.					Longwall	All coal removed.	
33	South Staffordshire								
34	Castleford Sewage Works	Shales and sandstones overlaid with marls and limestones.					Longwall	All coal removed.	
35	England	About 60 per cent. post; little loose soil or clay.				About level	Bord and pillar	About 50 per cent. of coal taken in 1st mining. Pillars about 30 ft. by 90 ft.	About 90
36	England	About 60 per cent. post; little loose soil or clay.	150 ft.	26°		8%	Bord and pillar	About 50 per cent. of coal taken in 1st mining. Pillars about 30 ft. by 90 ft.	
37	England	About 60 per cent. post; little loose soil or clay.	120 ft.	21°		6%	Bord and pillar	About 50 per cent. of coal taken in 1st mining. Pillars about 30 ft. by 90 ft.	
38	Yorkshire	2½ ft. strong blue bind; 45 ft. hard rock, ordinary coal measures; surface 8 ft. clay.	119 ft.	6° 25′		2.7%	Longwall	Area well packed.	100
39	Scotland		420 ft.			8.3%	Longwall	3 seams; total thickness 15 ft. had been worked out 20 yrs. before house was built.	

40			66 ft.						
41			180 ft. to 200 ft.						
42	Nuneaton		345 ft.						
43	Natal, South Africa	Sandstone and shales.		Over robbing 15 to 27°		20 to 33 %	Pillars 39 ft. by 75 ft.; places 15 ft. wide	Places kept over each other in both seams.	85
44	Lancashire, England	40 ft. to 60 ft. drift, surface nearly level. Strata about 80 per cent. shales, 20 per cent. sandstone.	80 ft.	5° ahead of vertical		16½°	Longwall	Gateways 9 ft. wide, 45 ft. centers. Area well packed.	100
45	Lancashire, England	40 ft. to 60 ft. drift, surface nearly level. Strata about 80 per cent. shales, 20 per cent. sandstone.	613 ft.	22°	At rise 25 ft. to 660 ft. deep	20°	Longwall	Gateways 9 ft. wide, 45 ft. centers. Area well packed.	100
46	Lancashire, England	40 ft. to 60 ft. drift, surface nearly level. Strata about 80 per cent. shales, 20 per cent. sandstone.	790 ft.	23°		18°	Longwall	Gateways 9 ft. wide, 45 ft. centers. Area well packed.	100
47	Lancashire, England	40 ft. to 60 ft. drift, surface nearly level. Strata about 80 per cent. shales, 20 per cent. sandstone.	750 ft.	19°		12°	Longwall	Gateways 9 ft. wide, 45 ft. centers. Area well packed.	100
48	Lancashire, England	40 ft. to 60 ft. drift, surface nearly level. Strata about 80 per cent. shales, 20 per cent. sandstone.	350 ft.		8° ahead of vertical	12°	Longwall	Gateways 9 ft. wide, 45 ft. centers. Area well packed.	100
49	Lancashire, England	40 ft. to 60 ft. drift, surface nearly level. Strata about 80 per cent. shales, 20 per cent. sandstone.	1290 ft.	20°		12°	Longwall	Gateways 9 ft. wide, 45 ft. centers. Area well packed.	

APPENDIX C.—*Data of Subsidence in Foreign Mines*—(*Continued*)

No.	Location	Authority	Remarks
1	South Kirby Colliery, Wales	Engineering Experiment Sta. No. 91. University of Illinois *Bull.* No. 49.	Part of bottom was removed. Maximum subsidence partly due to crushing of shaft pillar in seam above.
2	Derbyshire	*Idem.*	
3	South Waler	*Idem.*	
4	Bent Colliery, Lanarkshire	*Idem.*	(1)
5	Hickleton Main Colliery	*Trans.*, **46**, 21.	Total subsidence occurred 666 ft. back of face.
6	Teversal & Peasley Collieries	*Idem.*, **38**, 128.	Cracks opened when face was 190 ft. from wall; at maximum when face was under wall; closed afterward.
7	Stuffynwood Hall	*Idem.*, **36**, 427.	Settlement commenced when face was 300 ft. from building.[2]
8	Downhill Reservoir	University of Illinois *Bull.* 49.	Reservoir was over a barrier pillar between three mines.
9	Whettleford Pump Station	*Colliery Guard.*, Dec. 31, 1920.	
10	Robinson's End Reservoir	*Idem.*	
10-A	Robinson's Filters	*Idem.*	
11	Castleford, Yorkshire	*Idem.*, Jan. 14, 1921.	
12	India, Case No. 2	Report of Subsidence Committee, *Min. & Geol. Inst. of India.* Fig. 29.	Areas mined out: 650 by 340 ft.; 550 by 260 ft.[3]
13	India, Case No. 9	*Idem.* Fig. 27.	Area mined out: 430 by 280 ft.[3]
14	India, Case No. 16	*Idem.* Fig. 30.	
15	India, Case No. 15	*Idem.* Fig. 31.	Area mined out: 16 acres.[3]
16	Ayrshire	*Iron & Coal Trades Rev.*, Oct. 13, 1914.	Angle of fracture observed in 4 ft. seam 90 ft. above seam worked 1st slice.
17		*Colliery Engineer*, **2**, 25.	2nd slice. Observed by Fayol.
18	South Yorkshire	*Trans.* Inst. Min. Surveyors, **2**, 32.	
19	Doncaster	*Colliery Guardian*, Nov. 14, 1921.	Settlement measured while face was 100 ft. away.
20	South Yorkshire	*Idem.*	When face stopped, effect on surface was shown at ¼ to ⅙ of the depth in front of face.
21		*Idem.*, Mar. 18, 1921.	Width of strata in process of sinking, 40 ft.
22		*Idem.*	Width, 40 ft.
23		*Idem.*	Width, 70 ft.
24		*Idem.*	Width, 105 ft.
25		*Inst. Civ. Eng.* (1898) **135**, 116.	House stood on a fault against which workings stopped, pull followed line of fault.
26		*Idem.*, 117.	
27		*Idem.*, 117.	
28	Nottinghamshire-Newstead Abbey	*Idem.*, 127.	Fine building was not damaged.
29	Nottinghamshire	*Idem.*, 128.	No other damage done.
30	West of England	*Idem.*, 129.	
31	West of England	*Idem.*, 130.	
32	South Staffordshire	*Idem.*, 156.	
33	South Staffordshire	*Idem.*, 160.	Traffic was only interrupted twice.
34	Castleford Sewage Works	*Idem.*, 162.	
35	England	Jos. Eltringhem, Stud. I. M. E. Reinforcement of Buildings and Their Foundations Against Mining Subsidence, 1922.	
36	England	*Idem.*	Rate of pillar removal 5 ft. to 8 ft. per week.
37	England	*Idem.*	
38	Yorkshire	*Inst. Min. Eng.*, **57**, 74.	

39	Scotland	*Idem.*, **57**, 96.	Maximum subsidence possibly due to some settlement of upper seams.
40		*Idem.*, **28**, 320.	
41		*Idem.*, **28**, 320.	
42	Nuneaton	*Idem.*, **9**, 102.	Building was to dip of workings.
43	Natal, South Africa	*Colliery Guardian*, May, 19, 1922.	Angle of break never exceeded vertical over large areas.[4]
44	Lancashire, England	*Royal Comm. on Min. Subsidence*, Fig. 28.	Testimony T. A. O'Donohue, 6th day. Length of face about 900 ft.[5]
45	Lancashire, England	*Idem.*, Fig. 2.	Testimony T. A. O'Donohue, 6th day.
46	Lancashire, England	*Idem.*, Fig. 3.	Testimony T. A. O'Donohue, 6th day.
47	Lancashire, England	*Idem.*, Fig. 4.	Testimony T. A. O'Donohue, 6th day.
48	Lancashire, England	*Idem.*, Fig. 5.	Testimony T. A. O'Donohue, 6th day.
49	Lancashire, England	*Idem.*, Fig. 6.	Testimony T. A. O'Dononue, 6th day.

[1] The wave of maximum subsidence followed the working face at an average distance back of 186 ft., or 1 ft. horizontal for each 3½ ft. vertical, an angle of 16°. Maximum subsidence was approximately over center of excavated area.

[2] When the longwall face was 240 ft. from the hall, cracks on the surface were noted. Where the working face was almost vertically beneath, the cracks had attained their maximum width and thereafter began to close. When the face had advanced 300 ft. farther the walls of the building had assumed practically their normal position.

[3] In case 2, the greater depth produced a greater subsidence with a smaller mined out area. In case 9, two small faults have drawn subsidence over in their direction. In case 16, an area of 4 acres had been mined out without subsidence until an area on the higher side separated from the first by a thin barrier of pillars had been attacked. About 5 acres of the new area were extracted before subsidence occurred. Surface subsidence corresponded fairly closely with the original lower mined area. Maximum subsidence is over the center of lower area.

[4] In one case, a reservation around a building where the pillars were taken out all around, the rock break took the direct vertical line, but subsequently the thick bed of surface clay gradually travelled toward the gob edge, and in the course of several years there was a drop of about 4 in. and a lateral travel of about 3 inches for 70 ft. over the solid and 350 ft. over the coal.

[5] In no case did the full amount of subsidence extend beyond the edges of the workings. Depth of cover given is at point of full subsidence. In cases 44, 45, 46, the full subsidence occurs nearly 900 ft. behind the face of the workings which are on the rise side. With workings to the dip, the full subsidence in one case is 600 ft. behind the edge of the workings. In 47, 48, 49, with the workings to the dip, the full subsidence is 550 ft. behind the edge of the workings. Subsidence has been very regular; the comparatively few distinct breaks which have occurred are no doubt due to the thickness of the alluvial drift at the surface. Extensive buildings over the area have suffered little damage.

APPENDIX D.—*Data of Subsidence in Mines of United States*

No.	Location	Seam Worked	Thickness		Depth of Cover	Seam Thickness, Per Cent.	Subsidence			
			Total	Mined			Due to	Amount and Character	Began after Robbing	Ended
1	Franklin Co., Ill.	No. 6	9 ft. 10 in.	8 ft.	550 ft.		Squeezes	Cracks 8 in. to 10 in. and 2 in. to 3 in. wide.		
2	Carterville, Ill.		7 ft.	7 ft.	120 ft.		Falls	Holes 20 ft.; diam. 2 to 20 ft. deep.		
3	Perry Co., Ill.	No. 6	9 ft. 11 in.	6 ft. 11 in.	350 ft.		Squeezes	Sag 600 ft.; diam., max. depth 4 ft.	1 day	
4	Perry Co., Ill.	No. 6	9 ft. 11 in.	6 ft. 11 in.	350 ft.		Squeezes	Sag 800 ft.; diam., max. depth 3 ft.		
5	Franklin Co., Ill.	No. 6	9 ft.	8 ft.	450 ft.		Squeezes	Sags 18 in. to 4 ft. R. R. bridge dropped 18 in.; concrete walks broke.	2 to 3 mo.	6 mo.
6	Randolph Co., Ill.		6 ft.	6 ft.	150 ft.		Squeezes	Swamps formed; sag about 2 ft.		
7	Clifford Co., Ill.		7.5 ft.	7.5 ft.	140 ft.		Squeezes	Pond about 1 acre; max. depth 4 ft.		
8	Franklin Co., Ill.		8 ft.	8 ft.	460 ft.		Squeezes	Pond formed; sag about 2 ft.		
9	Vermilion Co., Ill.		6 ft.	6 ft.	210 ft.		Squeezes	Pond formed; max. sag 4.7 ft.		
10	Nokomis, Ill.		8 ft.	8 ft.	625 ft.		Squeezes	Pond formed.		
11	Nokomis, Ill.		7 ft.	7 ft.	330 ft.		Squeezes	Sags from 2 ft. to 3 ft. deep, fireclay bottom.	3 to 5 yr.	
12	Franklin Co., Ill.		9 ft.	9 ft.	425 ft.		Squeezes	Sag 4 ft. deep; R. R. track subsided 4 ft.		
13	Dist. No. 1, La Salle, Livingston, Grundy Counties.	No. 2	3 ft.	3 ft.	Up to 200 ft.		Pillar withdrawal	Subsidence averaged (55 per cent.) 1.7 ft.		
14	Dist. No. 1, La Salle, Livingston, Grundy Counties.	No. 2	3.3 ft.	3.3 ft.	200 ft. to 400 ft.		Pillar withdrawal	Subsidence averaged (50 per cent.) 1.7 ft.		
15	Dist. No. 1, La Salle, Livingston, Grundy Counties.	No. 2	3.2 ft.	3.2 ft.	400 ft. to 500 ft.		Pillar withdrawal	Subsidence averaged (39 per cent.) 1.2 ft.		
16	Georges Creek, Md.	Pittsburgh	6.5 ft. to 9.8 ft.	6.5 ft. to 9.8 ft.	250 ft.		Pillar withdrawal	Cracks.		
17	Georges Creek, Md.	Pittsburgh	8 ft.	8 ft.	170 ft.		Pillar withdrawal	Cracks		
18	Southwest Virginia.	A	5 ft. 10 in.	5 ft. 10 in.	570 ft. to 794 ft.		Pillar withdrawal	Cracks, 8 in. to 18 in. wide; no displacement.		
19	Connellsville Region, Pa.	Pittsburgh	8 ft.	7 ft.	178 ft.	27	Pillar withdrawal	Subsidence 24 in.	About 3 mo.	About 18 mo.
20	Connellsville Region, Pa.	Pittsburgh	8 ft.	7 ft.	435 ft.		Pillar withdrawal	Cracks, subsidence slight for 2 yrs. after pillars were removed.[1]	About 6 mo.	About 6 yr.
21	Illinois	No. 6	8 ft. to	7 ft. to	487 ft.	79	Squeeze	Crack on surface 6 in.	At once	About

			12 ft.	10 ft.				wide; subsidence from 0 ft. to 7.9 ft.		5 yr.
22	Utah....................		20 ft. to 24 ft.	20 ft. to 24 ft.	40 ft. to 80 ft.	84	Pillar withdrawal	Settlement from 6 ft. to 20 ft. deep.		
23	Connellsville Region, Pa....	Pittsburgh	8 ft.	8 ft.	381 ft. to 404 ft.	39	Pillar withdrawal	Cracks in pavements and houses.	At once	About 3¼ yr.
24	Southern West Virginia.....	Pocahontas No. 3	5 ft.	5 ft.	850 ft.		Pillar withdrawal	Surface cracks, about 1 ft. wide.		
25	Connellsville Region, Pa....	Pittsburgh	8 ft.	8 ft.	About 350 ft.		Pillar withdrawal	Buildings cracked; not over coal.		
26	Southern West Virginia.....	Beckley	8 ft. 6 in.	7 ft. 9 in.	About 500 ft.		Pillar withdrawal	Cracks on surface.		
27	Coal City, Ill..............		3 ft.		About 125 ft.		Longwall	House corner subsided 1 ft. on solid coal.		
28	Danville, Ill...............		6 ft.		About 200 ft.		Mine workings	House badly damaged.		
29	Springfield, Ill.............		5 ft. 9 in.		About 200 ft.		Mine workings	Houses cracked.		
30	Southern Illinois...........		8 ft.		About 450 ft.	37	Mine workings	Houses cracked, sag about 3 ft. deep.		
31	Eastern Oklahoma..........	Henryetta	3 ft.	3 ft.	About 165 ft.		Mine workings	School house was badly damaged; no appreciable subsidence.	Subsidence started 5 years after.	
32	Eastern Oklahoma (Adamson).	Lower Hartshorne	5 ft.	5 ft.	About 720 ft.		Squeeze	Ground broken by cracks, no appreciable subsidence.		

APPENDIX D.—*Data of Subsidence in Mines of United States*—(*Continued*)

No.	Location	Character of Strata over Coal	Dip of Strata	Angle of Draw: Over Robbing	Angle of Draw: Ahead of Robbing	Mining Methods: Kind	Mining Methods: Spacing Rooms, etc.	Coal Removed, Per Cent.
1	Franklin Co., Ill.	Shales, sandstone, limestone.				Room and pillar	40 ft. centers, 22 ft. wide.	67
2	Carterville, Ill.	Slate and clay.				Room and pillar		
3	Perry Co., Ill.					Room and pillar		49 to 61
4	Perry Co., Ill.					Room and pillar		49 to 61
5	Franklin Co., Ill.	Shales, sandstone, limestone, fireclay bottom.				Room and pillar	45 ft. centers, 250 ft. long, 30 ft. wide.	
6	Randolph Co., Ill.					Room and pillar	50 ft. centers, 32 ft. wide.	
7	Clifford Co., Ill.	Considerable quicksand.				Room and pillar	40 ft. centers, 22 ft. wide.	
8	Franklin Co., Ill.	Shales, sandstones, limestone.				Room and pillar	45 ft. centers, 30 ft. wide.	
9	Vermilion Co., Ill.					Room and pillar		
10	Nokomis, Ill.					Room and pillar		
11	Nokomis, Ill.	Fireclay bottom.				Room and pillar	60 ft. centers, 30 ft. wide.	60 to 64
12	Franklin Co., Ill.	Shales, sandstone, limestone.				Room and pillar	50 ft. centers, 300 ft. long, 30 ft. wide.	
13	Dist. No. 1, La Salle, Livingston, Grundy Counties.	Shales and sandstone.				Longwall		
14	Dist. No. 1, La Salle, Livingston, Grundy Counties.	Shales and sandstone.				Longwall		
15	Dist. No. 1, La Salle, Livingston, Grundy Counties.	Shales and limestone.				Longwall		
16	Georges Creek, Md.			20° 31′		Room and pillar		90
17	Georges Creek, Md.			14°		Room and pillar		90
18	Southwest Virginia	Sandstones and shales.		8° 35′ to 10° 23′		Room and pillar		85 to 90
19	Connellsville Region, Pa.	Sandstones and shales.						90
20	Connellsville Region, Pa.	Sandstones and shales.			0° 20′ to 1° 30′	Room and pillar	Buildings on a solid block of coal reserved.	90
21	Illinois	Sandstones and shales; occasional fire clays.				Room and pillar	Rooms 40 ft. centers, about 24 ft. to 30 ft. wide.	79
22	Utah	Sandstones.		0°		Room and pillar		90
23	Connellsville region, Pa.	Sandstones.		0°		Room and pillar	Rooms 90 ft. centers, 11 ft. wide.	90
24	Southern West Virginia	Sandstones.				Room and pillar		90
25	Connellsville Region, Pa.	Sandstones.				Room and pillar		
26	Southern West Virginia	Sandstones		3° 15′		Room and pillar		90
27	Coal City, Ill.				Yes	Longwall		
28	Danville, Ill.					Room and pillar		About 80
29	Springfield, Ill.					Room and pillar		
30	Southern Illinois					Room and pillar	Rooms 45 ft. centers, 30 ft. wide.	About 65

31	Eastern Oklahoma............	Mostly shale and thin beds of sandstone.				Room and pillar	Rooms 40 ft. to 50 ft. wide, very thin pillars.	About 68
32	Eastern Oklahoma (Adamson).	Mostly shale and thin beds of sandstone.	28°		250 ft. ahead of vertical line from face.	Room and pillar	Seam dipped 28° at face. Rooms 32 ft. centers, 24 ft. wide.	About 69

APPENDIX D.—*Data of Subsidence in Mines of United States*—(*Continued*)

No.	Location	Authority	Remarks
1	Franklin Co., Ill.	Illinois State Geol. Surv. *Bull.* No. 17, 33.	Pillars not removed.
2	Parterville, Ill.	*Idem.*, 37.	Pillars not removed.
3	Perry Co., Ill.	*Idem.*, 43.	Pillars not removed.
4	Perry Co., Ill.	*Idem.*, 44.	Pillars not removed.
5	Franklin Co., Ill.	*Idem.*, 44.	After heavy falls surface subsidence showed in 12 hr. Movement began 2 to 3 mo. after rooms were finished. Continued in mine for several weeks.
6	Randolph Co., Ill.	*Idem.*, 47.	Pillars not removed.
7	Clifford Co., Ill.	*Idem.*, 48	Pillars not removed.
8	Franklin Co., Ill.	*Idem.*, 48.	Pillars not removed.
9	Vermilion Co., Ill.	*Idem.*, 49.	Pillars not removed.
10	Nokomis, Ill.	*Idem.*, 49.	Pillars not removed.
11	Nokomis, Ill.	*Idem.*, 58.	Pillars not removed.
12	Franklin Co., Ill.	*Idem.*, 62.	Pillars not removed. Squeezed area 1200 ft. square. Fireclay bottom.
13	Dist. No. 1, La Salle, Livingston, Grundy Counties.	*Idem.*, 77.	Data from 10 mines; not sufficient to warrant these figures as basis of estimate of general subsidence in long-wall district.
14	Dist. No. 1, La Salle, Livingston, Grundy Counties.	*Idem.*, 77.	
15	Dist. No. 1, La Salle, Livingston, Grundy Counties.	*Idem.*, 77.	
16	Georges Creek, Md.	Univ. of Illinois *Bull.* No. 49.	
17	Georges Creek, Md.	Univ. of Illinois *Bull.* No. 49.	
18	Southwest Virginia	Committee on Ground Movement and Subsidence. Fig. 20.	Cracks were parallel to starting line of robbing and about 1200 ft. long.
19	Connellsville Region, Pa.	*Idem.*, Fig. 8.	Pillars partly taken out under a barn. Apparent settlement of 10 in. of house 100 ft. from pillars on solid coal.
20	Connellsville Region, Pa.	*Idem.*, Fig. 9. *Idem.*, Table 1.	(1)
21	Illinois	*Idem.*, Fig. 25.	(2)
22	Utah	*Idem.*, Fig. 26.	
23	Connellsville Region, Pa.	*Idem.*, Fig. 13.	(3)
24	Southern West Virginia	*Idem.*, Fig. 21	Amount of subsidence not noted.
25	Connellsville Region, Pa.	*Idem.*, Fig. 63.	(4)
26	Southern West Virginia	*Idem.*, Fig. 21.	(5)
27	Coal City, Ill.	Illinois Geol. Surv. *Bull.* No. 17, 15.	
28	Danville, Ill.	*Idem.*, 66.	Pillars not removed, but practically all coal gone.
29	Springfield, Ill.	*Idem.*, 66.	Pillars not removed, but most of coal gone.
30	Southern Illinois	*Idem.*, 67.	
31	Eastern Oklahoma	J. J. Rutledge: *Trans.* (1923).	Pillars not removed; practically all coal gone. Building of brick, poorly built, cracked by horizontal pull mainly.
32	Eastern Oklahoma (Adamson)	*Idem.*	(6)

[1] Two houses and a shop were on a reservation in the shape of an isosceles triangle with a base 192 ft. long and an altitude of 244 ft., in which all of coal was left. Nearest point of either house was 18 ft. from line of pillar work. About 6 mo. after coal was removed, although only a strip 20 ft. wide was taken out along northern side, cracks developed in each building and a surface crack about 40 ft. long, varying from 2 ft. to 10 ft. from line of pillars, showed over the solid coal. Levels were taken on the foundations, beginning about a year after coal was removed and continued at monthly intervals for 20 mo., during which time settlement did not exceed 0.2 ft. at any point and was usually much less than this; considering the accuracy of the work it was practically nothing. Levels were taken again after 14 mo. with little change noticed. About 19 mo. after this, or about 5¼ years after all the coal was removed around the reservation, serious surface cracks, in places 2 ft. to 4 ft. wide, developed over the solid coal, leading from cracks over the mined out area to a point 50 ft. from the pillar line, and at the same time settlement of the buildings, ranging from 0.4 ft. to 0.7 ft., occurred. This was probably caused by movement due to robbing north of *narrow strip along northern side of block left.* The reservation was on top of a hill, from which the ground sloped in all directions and evidently this subsidence was due to the settlement of the surface clay and dirt toward the mined out areas.

[2] Over this area the surface subsided in a saucer shape, 7.9 ft. deep in the middle and slight subsidence extended over the mined out area. A crack 6 in. wide showed on the surface. No pillars were drawn. Fireclay bottom under coal is 1 ft. thick. Seventy-nine per cent. of coal was removed from the mined over area. Bottom did not heave.

[3] In a portion of this area, under a street, pillars were left to support the street, over an area 175 ft. long by 80 ft. wide, 39 per cent. of the coal being left in place and the pillars being drawn on both sides. These pillars squeezed and subsidence over them varied from nothing to a maximum of 1.3 ft., at a depth of 384 ft., the greater subsidence being 15 per cent. of seam thickness.

[4] Two brick churches were situated on separate reservations, around each of which all coal had been removed. Pillars were removed to within 20 ft. of one building and 30 ft. of the other. No ground cracks were observed but walls of each building were cracked and some plaster had fallen off. In one case this damage was at least partly, and probably wholly, due to poor workmanship, this building being the second one mentioned. If these cracks are due to subsidence, they must be due to movement of the surface soil alone, as no disturbance of the ground was noted over the mined out areas.

[5] No measurements were made of subsidence, but several large surface cracks developed on the hillsides. In the bottom where the cover varied from 440 ft. to 460 ft., all the coal was taken out from under a large creek for a distance of 1200 ft. without apparently causing much increase of water in the mine. The surface soil is here probably 30 ft. thick.

[6] Mine suddenly collapsed, without warning, entombing 13 men. Cave began about 370 ft. from outcrop, under 200 ft. cover, and extended 1440 ft. along slope to face, under 720 ft. cover. No levels were taken of amount of subsidence, but workings were under the town and streets were shown on mine map, so cracks were accurately located. While extreme crack was 250 ft. ahead of a vertical line at the face, or an angle of 19°, it was 150 ft. back of a normal line to the seam at the face, or 9°. Only about 31 per cent. of the coal had been left unmined, and it is remarkable the squeeze did not occur sooner, under such heavy cover.

APPENDIX D.—*Data of Subsidence in Mines of United States*—(*Continued*)

No.	Location	Seam Worked	Thickness		Depth of Cover	Due to	Subsidence			
			Total	Mined			Amount and Character	Thickness of Seam, Per Cent.	Began after Robbing	Ended
33	Connellsville Region, Pa.	Pittsburgh	8 ft. 0 in.	7 ft. 6 in.	725 ft.	Pillar withdrawal	Buildings were thrown out of plumb. Brick paving of street had to be replaced. Subsidence 2.8 ft.	27	At once.	2 yr.
34	Illinois	No. 6	7 ft. 0 in.	7 ft. 0 in.	350 ft.	Squeezes	Max. 2 ft.	27		
35	Alabama	Mary Lee	6 ft. 5 in.	6 ft. 5 in.	450 ft.	Pillar withdrawal	Cracks 4 in. wide.		After large falls are made	
36	Colorado	Northern-lignite	10 ft. to 11 ft.	8 ft.	150 ft.	Pillar withdrawal	Max. 2 ft.	25	After falls are made	Continuing
37	Colorado	Northern-lignite	10 ft.	8 ft.	100 ft.	Pillar withdrawal	Max. 5 ft.	62	2 to 3 days after falls	
38	Allegheny County, Pa.	Upper Freeport	6 ft. 6 in.	6 ft. 6 in.	140 ft.	Pillar withdrawal	Subsidence not measured; cracks.		Very soon after falls	
39	Cambria County, Pa.	"C"	4 ft. 6 in.	4 ft. 6 in.	220 ft.	Pillar withdrawal	House over solid coal adjoining was cracked 2 in.; no subsidence measured.			
40	Christian County, Ill.	No. 6	7 ft. 6 in.	7 ft. 6 in.	328 ft.	Squeezes	Max. about 18 in.	20	Before panels are driven up	None after first drop
41	Washington County, Pa.	Pittsburgh	5 ft.	5 ft.	115 ft.	Cracks over solid coal				
42	Somerset County, Pa.	Pittsburgh	8 ft.	8 ft.	26 ft. to 35 ft.	Pillar withdrawal	Maximum 3 ft. in seam above.	38		
43	Frostburg, Md.	Pittsburgh	14 ft.	8 ft. to 9 ft.	About 112 ft.	Pillar withdrawal	Maximum 1.5 ft. in seam above.	18		
44	Somerset County, Pa.	"B"	4 ft.	4 ft.	84 ft.	Pillar withdrawal	Maximum 0.7 ft. in seam above.	18		
45	Fayette County, Pa.	Pittsburgh	8 ft.	7 ft.	80 ft.	Pillar withdrawal	Maximum 4.5 ft. in seam above.	64		
46	Cambria County, Pa.	"C"	5 ft. 3 in.	5 ft. 3 in.	25 ft.	Pillar withdrawal	"D" seam badly crushed and misplaced.			
47	Cambria County, Pa.	"C"	5 ft. 3 in.	5 ft. 3 in.	120 ft.	Pillar withdrawal	"E" seam broken; cracks 3 in. to 4 in. wide.			
48	Somerset County, Pa.	Pittsburgh	8 ft.	8 ft.	25 ft.	Pillar withdrawal	Max. 4 ft.	50		
49	McDowell County, W. Va.	No. 3	7.1 ft.	7.1 ft.	310 ft.	Pillar withdrawal	Cracks ¼ in. to 3 in. wide.			

50	Allegany Co., Maryland..	Pittsburgh	12 ft. to 14 ft.	Usually 8 ft. to 9 ft.	92 ft. between seams	Mine workings	Tyson seam subsided 0.7 ft.	8		
51	Allegany Co., Maryland..	Pittsburgh			96 ft. between seams	Mine workings	Tyson seam cut out entirely.			
52	Allegany Co., Maryland..	Pittsburgh			101 ft. between seams	Mine workings	Tyson seam subsided 1 ft.; bottom of coal dropped away.	12		
53	Somerset Co. Pa.........	Redstone	8 ft.	8 ft.	25 ft. between seams	Mine workings	Redstone seam subsided from 3 ft. to 4 ft.	44		
54	Maryland...............	Pittsburgh	8 ft.	8 ft.	About 160 ft.	Pillar withdrawal	Crack showed in dam and some water entered mine.			
55	Maryland...............	Pittsburgh		8 ft. 5 in.	About 180 ft.	Pillar withdrawal	Ground broken by cracks; subsidence 5 ft.	59		
56	Maryland...............	Pittsburgh		8 ft. 5 in.	About 170 ft.	Pillar withdrawal	Ground broken by cracks; 6 in. to 12 in. wide.			
57	Maryland...............	Pittsburgh		8 ft.	96 ft.	Mining	Break found in Sewickley seam and ribs crushed.			
58	Maryland...............	Pittsburgh and Sewickley		8 ft. and 3.5 ft.	About 140 ft.	Mining	Cracks in surface.			
59	Fayette Co., Pa..........	Pittsburgh	7.5 ft.	7.5 ft.	About 420 ft.	Pillar withdrawal	Cracks 2 ft. to 5 ft. wide; no levels taken.			
60	Allegheny Co., Pa........	Pittsburgh	6.5 ft.	6.5 ft.	220 ft.	Pillar withdrawal	Cracks 2 in. wide; no levels taken.			
61	Westmoreland Co. ,Pa....	Pittsburgh	7.5 ft.	7.5 ft.	245 ft.	Pillar withdrawal	Brick building subsided from 5 in. to 17 in.	17	1 mo.	Not ended 28 mo.
62	Allegheny Co., Pa........	Upper Freeport	6.5 ft.	6.5 ft.	575 ft.	Pillar withdrawal	Cracks 6½ ft. wide; subsidence 1.6 ft.	25	About 2 mo. after fall	
63	Greene Co., Pa...........	Pittsburgh	8.0 ft.	8.0 ft.	520 ft.	Pillar withdrawal	Cracks 5 ft. to 7 ft.; max. subsidence 1.5 ft.			
64										

APPENDIX D.—*Data of Subsidence in Mines of United States*—(*Continued*)

No.	Location	Character of Strata over Coal	Distance Ahead	Angle Ahead	Angle Back	Distance Back	Dip of Strata	Mining Methods		Coal Removed, Per Cent.
								Kind	Spacing Rooms, Etc.	
33	Connellsville Region, Pa.	Shales; sandstones.					5%	Room and pillar	Rooms 12 ft. wide, 80 to 100 ft. centers.	90
34	Illinois	0 to 5 ft. shales under 20 ft. limestone; bottom 12 in. fireclay, heaves under weight.						Room and pillar	Rooms 30 ft. wide, 70 ft. centers.	
35	Alabama	Shales and sandstones; roof breaks after about 2 acres are mined out.				Always over mined area	4%	Room and pillar	Rooms 28 ft. wide, 58 ft. centers.	
36	Colorado	Soft sandstone and clay; breaks over areas 40 ft. by 125 ft.				Vertical	2.2%	Room and pillar	Rooms 22 ft. wide, 45 ft. centers.	
37	Colorado	Soapstones, sand, slates and soft sandstones; easily broken.						Room and pillar	Rooms 20 ft. wide, 40 ft. centers.	95
38	Allegheny Co. Pa.	Slate, shales and sandstone; breaks after about 2.2 acres are mined out.			20°	50 ft.	1.5%	Room and pillar	Rooms 25 ft. wide, 50 ft. centers.	
39	Cambria Co. Pa.	Shale and fireclay; about 90 ft. clay and boulders.	73 ft.	19° 45′			About level	Room and pillar	Rooms 27 ft. wide, 61 ft. centers.	90
40	Christian Co. Ill.	Shales, sandstone and limestone. Fire-clay bottom.				Over mined out area	About level	Room and pillar	Rooms 30 ft. wide, 50 ft. and 60 ft. centers.	52 to 65
41	Washington Co. Pa.	Shales, limestone and sandstone.	40 ft. 104 ft.	18° 37°			About 2%	Room and pillar	Rooms 20 ft. wide, 38 ft. centers.	
42	Somerset Co. Pa.	Slates.				Always over mined out area	About 2%	Room and pillar		90
43	Frostburg, Md.	Shales, slate and 16 ft. sandstone.				Always over mined out area	About 2%	Room and pillar		90
44	Somerset Co. Pa.	Shales and slate.				Always over mined out area	About 2%	Room and pillar		90
45	Fayette Co. Pa.	Shales and slate.				Always over mined out area	$3\frac{1}{2}$%	Room and pillar		90
46	Cambria Co. Pa.	Shales, slate and sandstone.				Always over mined out area		Room and pillar		
47	Cambria Co. Pa.	Shales, slate and sandstone.				Always over mined out area				
48	Somerset Co., Pa.	2 ft. to 5 ft. sandstone; balance soft slate and shales.				Always over mined out area				

49	McDowell Co., W. Va.....	Sandstones and slates.					About 3%	Room and pillar		74
50	Allegany Co., Maryland...	Shales and some sandstone.								
51	Allegany Co., Maryland...	Shales and some sandstone.								
52	Allegany Co., Maryland...	Shales and some sandstone.								
53	Somerset Co., Pa..........	5 ft. hard sandstone on top of Pittsburgh seam; balance soft slates and shales.								
54	Maryland................	Shales and some sandstone.					About 3%	Room and pillar		
55	Maryland................	Shales and some sandstone.	About 10° over robbing					Room and pillar	Rooms 14 to 75 ft. centers.	About 85
56	Maryland................	Shales and some sandstone.	11° to 20° over robbing	30′ in 170′ About 10° ahead of robbing						About 85
57	Maryland................	Shales and some sandstone.						Room and pillar		About 85
58	Maryland................	Shales and some sandstone.						Room and pillar		About 85
59	Fayette Co., Pa...........	Shales and sandstones.	3° to 12° over robbing					Room and pillar		About 85
60	Allegheny Co., Pa.........	Sandstones and shales.	8° about 30′				1.5%	Room and pillar		About 60
61	Westmoreland Co., Pa.....	Sandstones and shales.	12° about 50′				2%	Room and pillar		90
62	Allegheny Co., Pa.........	Sandstones and shale. About 4 ft. surface soil.	20 ft.	2° 15′			About level	Room and pillar	Rooms 20 to 70 ft. centers.	
63	Greene Co., Pa............	Sandstones and shale.				95 ft. to 150 ft.	2%	Room and pillar	Rooms 10 to 100 ft. centers.	
64										

APPENDIX D.—*Data of Subsidence in Mines of United States*—(*Continued*)

No.	Location	Authority	Remarks
33	Connellsville Region, Pa.	Committee on Ground Movement and Subsidence. Correspondence. Plan 6–283.	Will be completed later.
34	Illinois	*Idem.*	
35	Alabama	*Idem.*	
36	Colorado	*Idem.*	
37	Colorado	*Idem.*	
38	Allegheny Co., Pa.	*Idem.*	
39	Cambria Co., Pa.	*Idem.*, Fig. 12.	Gas and water lines on solid coal were pulled apart 4 to 6 in.
40	Christian Co., Ill.	*Idem.*, Fig. 24.	(1)
41	Washington Co., Pa.	*Idem.*, Fig. 10.	(2)
42	Somerset Co., Pa.	*Idem.*, Fig. 1.	Subsidence noted when bottom seam is worked out first.
43	Frostburg, Md.	*Idem.*	Subsidence noted when bottom seam is worked out first.
44	Somerset Co., Pa.	*Idem.*	Subsidence noted when bottom seam is worked out first.
45	Fayette Co., Pa.	*Idem.*, Fig. 3.	Subsidence noted when bottom seam is worked after upper seam workings were driven.
46	Cambria Co., Pa.	*Idem.*, Fig. 5.	(3)
47	Cambria Co., Pa.	*Idem.*, Fig. 5.	(4)
48	Somerset Co., Pa.	*Idem.*, Fig. 2.	
49	McDowell Co., W. Va.	*Idem.*, Fig. 22.	(5)
50	Allegany Co., Maryland	*Idem.*, Fig. 19.	Seam below had been worked years ago; condition of workings unknown.[6]
51	Allegany Co., Maryland	*Idem.*, Fig. 19.	Seam below had been worked years ago; condition of workings unknown.[6]
52	Allegany Co., Maryland	*Idem.*, Fig. 19.	Bottom of seam 1 ft. thick dropped away, top part apparently undisturbed. (See Case 50, "Remarks.")
53	Somerset Co., Pa.	*Idem.*	Condition of old workings unknown; seam and strata above have subsided.[6]
54	Maryland	*Idem.*, Fig. 17.	(7)
55	Maryland	*Idem.*, Fig. 18.	
56	Maryland	*Idem.*, Fig. 18.	(8)
57	Maryland	*Idem.*, Fig. 15.	(9)
58	Maryland	*Idem.*, Fig. 16.	(10)
59	Fayette Co., Pa.	*Idem.*, Fig. 4.	
60	Allegheny Co., Pa.	*Idem.*, Fig. 7.	(11)
61	Westmoreland Co., Pa.	*Idem.*	Will be completed later.[12]
62	Allegheny Co., Pa.	*Idem.*, Fig. 14.	Crack 750 ft. long of which 310 ft. was over solid coal. Total length of pillar line 1600 ft.
63	Greene Co., Pa.	*Idem.*, Fig. 11.	
64		7–49.	Will be completed later.

[1] Where the bottom clay is over 2 ft. thick, and gets wet, the squeezes usually occur before the end of the panel is reached, panels being 900 to 1500 ft. ong. In other places, squeezes do not occur until after work in panel is completed, if then. After first subsidence is noted, no appreciable movement is noticed, although no levels are taken. On good bottom, the same amount of coal left in place supports the surface, and this amount is evidently about the minimum that can safely be left.

[2] Coal was left to support a house. Pillars were removed to within 100 ft. of nearest point of house, and the crack developed on the surface as shown, over the solid area. House was on a slope and direction of pillar removal was uphill.

[3] Working in the "D" seam had to be abandoned on account of crushing and misplacing of coal, and bad roof and bottom, due to pillar removal in "C" seam below.

[4] Workings in the "E" seam require more timber and top falls easier over mined out areas than elsewhere. Coal makes more slack and is more easily pick-mined under same conditions.

[5] Aside from the cracks noted, no evidences of subsidence were found. Water usually remains in upper seam, and will not drain to lower one. Surface cracks have been found 6 in. to 8 in. over the upper seam, due to mining in the bottom one.

[6] In cases 50, 51, 52 and 53, the mines in the Pittsburgh seam had been worked many years ago, the extraction not having been complete as most of the workings in that seam now are confined to recovering old pillars. It cannot be told now how much of the trouble in the upper seam is due to pillar withdrawal and how much to falls, due to inadequate pillars; apparently a very large part of it is due to the latter cause.

No. 51 is interesting as it is the only case in a large experience with this class of mining where the upper seam has been found to have dropped entirely out of place.

[7] The dam is a cut stone structure about 260 ft. long, probably 18 ft. high at maximum and is founded on rock. All of the coal was removed from under the dam and reservoir, and under the stream (a fairly large one), for a distance of about ¾ mile. The dam breast was cracked somewhat, but without causing any material damage. Some water entered the mine through cracks and caused considerable extra pumping, but it was finally stopped by putting straw into the stream, which found its way into the cracks and they were soon filled with silt. At the upper end of the dam, and with about 200 ft. cover, the same pillar withdrawal made cracks 3 in. to 6 in. wide in a rock cut on an electric railway line.

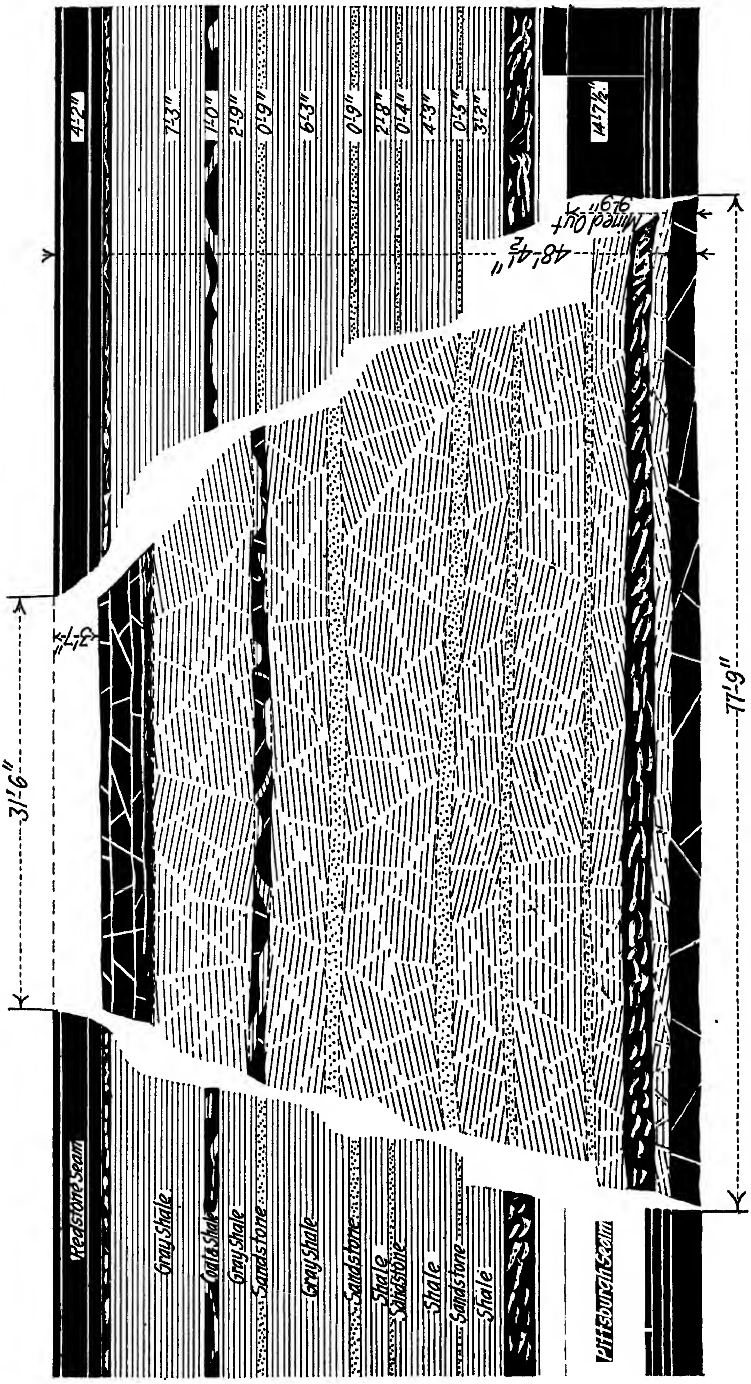
[8] The rock cuts on the railroad lines showed cracks 3 in. to 6 in. wide. The ground between the railroad and the creek has a covering of about 30 ft. of soil, which is partly why the cracks extend beyond the worked out area at room 4. Where pillar withdrawal began the cracks are well within the worked out area.

[9] The condition of the old workings in the lower (Pittsburgh) seam is not known, but evidently they are not as shown, as the conditions found in the upper (Sewickley) seam could hardly have been caused by mining only the rooms, as shown below.

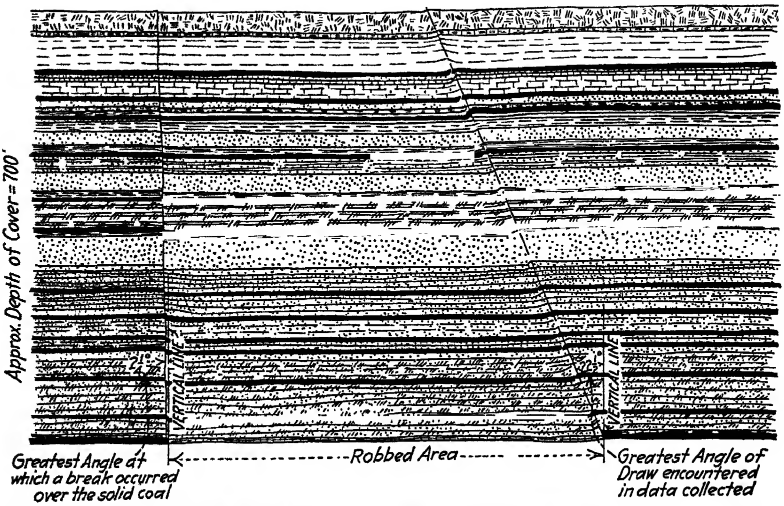
[10] In this case the workings in the upper (Sewickley) seam had been abandoned for some years, after some of the pillars had been drawn and a part of it left in stumps. About nine years later workings in the lower (Pittsburgh) seam reached this area and evidently disturbed the coal left in the Sewickley seam, causing the crack on the surface from A to B.

[11] Stumps of coal were left, as shown, between A and B to support paved road. About four years later, when pillars were withdrawn adjoining these stumps, the crack in the surface developed. Evidently the movement of strata due to pillar work caused the break extending over the solid coal.

[12] In this case a brick church was built on an area from under which all coal had been mined about five years before, the nearest of solid coal being 50 ft. from end of building. Fifteen years later, and 20 years after the original mining was done, pillar work was commenced in the adjoining areas and continued to date. Shortly after it began the building began to subside and has not yet come to a stop. The movement of the strata over the adjoining area evidently started the movement of the apparently settled area under the church.



APPENDIX E.—TRUE SECTION SHOWING THE ACTION OF OVERLYING STRATA ON REMOVAL OF PILLARS IN THE PITTSBURGH OR "BIG VEIN" SEAM, GEORGES CREEK REGION, MARYLAND.



APPENDIX F.—PROBABLE MOVEMENT OF STRATA OVER ROOM AND PILLAR MINING FROM WHICH ALL PILLARS HAVE BEEN WITHDRAWN.

DISCUSSION

H. G. MOULTON, New York, N. Y.—The work of the Subsidence Committee began in 1920 and a preliminary meeting was held in the spring of 1921. As comparatively little data existed, we realized that for a number of years the Committee would have to collect data of every conceivable nature with the thought that this information might serve as a basis for propounding new theories, for locating surface works, or for appraising damage to adjacent properties. In 1923, Rice's paper[5] set forth the underlying principles applicable to the subsidence of ground in the light of information then available and recommended the systematic collection of further data. As a result three sub-committees were appointed: (*a*) underground metal mining practice, (*b*) open-cut practices,[6] and (*c*) coal mining. The third sub-committee, under the leadership of Howard N. Eavenson, has presented data pertaining to bituminous coal mining operations. The anthracite section is now at work collecting data for the anthracite field.

A form of draw similar to that noted in longwall methods may be observed in metal-mining methods that involve continuous and complete withdrawal of a given area. In caving, the area of subsidence generally extends outside the area of extraction, which leads to the conclusion that the prime factor of importance is time for stresses to adjust themselves.

[5] G. S. Rice: Some Problems in Ground Movement and Subsidence. *Trans.* (1923) **69**, 374.

[6] Louis Cates: Factors Affecting Bank Slopes in Steam-shovel Operations. See page 818.

When an area is extracted by room-and-pillar, a small area is taken out and the ground readjusts itself to the stresses set up; another area is extracted and the ground is adjusted to that, until finally an arching effect takes place over a period of time, which could not have happened in a shorter period. The quick effect of the stresses set up by continuous removal through longwall is the controlling factor in the difference between the results of the two methods.

H. N. Eavenson.—That point has been brought out in a new way by machine mining. Where machines have been used extensively for loading coal where the top is bad, much less trouble with the top occurs as less time is needed to get out the coal. In faces of 130 to 140 ft., in workings tested in the Pittsburgh seam, the top was ordinarily classed as very bad and required much timbering, but when one of those faces moved rapidly enough, the top was not very bad and required very little timbering. Apparently they got the coal out before the stresses started. In such workings where long faces are used, I think that surface subsidence would act as it does over longwall workings.

H. G. Moulton.—The time factor in all operations involving ground movement or subsidence is one phase on which we know comparatively little now and on which our knowledge in the future may guide our mining methods to a much larger extent. We would recommend consideration of the effect of time, which again is a factor of the speed of extraction as well as of the methods used.

S. A. Taylor, Pittsburgh, Pa.—That big fall on the thick vein of Freeport near Harmonville, Pa., is one of the most remarkable examples I have seen. About 3 months ago they told me another crack was appearing out toward the river, making three large cracks over the solid coal.

T. G. Fear, Fairmont, W. Va.—We intend to start work in 30 days, using the modified longwall method and conveyors. There are 91 oil and gas wells in the 2100-acre tract, so that we must find out how the strata above the coal is going to act and what the angles of fracture are going to be, so as to leave enough wall around the active oil and gas wells. There seems to be very little data on this subject but soon we will have to know something about this.

G. S. Rice.—In my early work in the West, we were careful in the purchase of land always to include a clause that at least gave us a fighting chance so as not to be compelled to support the surface. Then, though, we might have had to pay some damages it was not in excess of the real damage.

One reason for obtaining precise data on subsidence is its help in selecting a mining method. When in Nova Scotia recently studying the cause and preventions of "bumps," I would have been very glad to have

had some knowledge of how the strata over the mine was breaking. The Empire Steel Co. has promised to put in lines of monuments over many of its principal workings, including longwall.

Eugene McAuliffe, of the Union Pacific Coal Co., with the aid of the Bureau of Mines' engineers, has laid out a plot and has agreed that his company will put in monuments and provide periodic surveys to obtain subsidence data. So much material could easily be obtained by mine operators if it was only taken in time.

The Bureau of Mines expects soon to publish some very definite and interesting information on longwall mining in Illinois. Longwall mining in northern Illinois is not very prosperous. When I was mining up there, we got out 6,000,000 tons a year; last fall the production was about 1,700,000 tons and in two of the veins the mines were apt to be closed. But longwall mining is returning to favor, through what is virtually longwall retreating.

S. A. Taylor.—About 25 years ago, I read an English work on subsidence that said that the overburden would not be affected when its thickness was ten times the thickness of the seam of coal, but I have seen a number of cases where the ground movement was much greater. One peculiar case was in the Pittsburgh seam near Monongahela City on the Monongahela River. The Sewickley seam at that point came in about 80 ft. above the Pittsburgh coal, and there was 35 to 40 ft. of shale. The Pittsburgh sandstone was not in its usual place, but in its place was shale. There was no subsidence whatever in the Sewickley coal when we started to mine it and as far as we had mined, the Sewickley seam was not interfered with whatever. I attributed that largely to the fact that the shale, instead of rock, overlaid the coal. Having found a good straight face on the shale, I cut out a portion carefully and found that 18 cu. ft. of the solid shale, dug out with a pick, would fill a cubic yard box. Of course, that was not conclusive as to what was happening in the mine, but it indicated somewhat the arching effect; or rather, the falling of the shale arched itself before it got up to the 80 ft. If the men making observations on a fall will at the same time note the character of the overlying strata, some prediction could be made as to where the falls are going to take place and how high they will go.

H. G. Moulton.—Crane[7] has said that roof breakage might be induced by blasting, also that special pillars might be built to receive roof falls and break them so that the swelling of ground might be made to bring about exactly the advantage that you have in mind, and in that way check the subsidence.

J. A. Garcia, Chicago, Ill.—Recently I had occasion to investigate the results of impact on the strata from shooting. Two operators were

[7] W. R. Crane: Roof Support in the Red Ore Mines of the Birmingham District. *Trans.* (1925) **72**, 187.

working, one in an upper seam and one in a lower, about 100 ft. apart. The lower seam was 5 ft. thick, the upper about 7 ft. The lower seam was a shaft mine, the upper a drift. The upper workings suddenly caved in and, shortly after, a squeeze developed in the lower mine. The operator of the upper seam sued the operator of the lower, claiming that the squeeze caused his workings to cave and the operator of the lower seam sued the other operator, claiming that the heavy shooting in the upper seam brought on the squeeze in the lower.

A fault plane cut from the surface straight down through both seams of coal and adjacent to the squeeze. A shot in the upper seam, would blow out pitcap lights in the lower mine and cause pieces of coal to fall off the ribs and slate from the roof. The operator of the lower seam based his claim on these facts. My investigation indicated that the shooting had some effect on the pillars and roof though I cannot say positively that the squeeze was brought on by shooting, for when the heavy charges of dynamite were shot the whole structure in lower workings would tremble.

C. M. Young, Lawrence, Kan.—If the roof shatters in falling so that its volume increases enough to compensate for the height of the bed taken out, the subsidence cannot be apparent on the surface. If it does not, subsidence on the surface is inevitable. On the other hand, if the roof practically does not break but bends (of course, we know that this bending is a series of breaks, as in the longwall system), then subsidence on the surface is inevitable, even though it is never noticed, as in the case in northern Illinois. It was thought there had been no subsidence because no one had observed it.

R. D. Hall, New York, N. Y.—Choking of the measures by falls appealed strongly to earlier investigators of the subject, and some people still declare that the roof falls until it chokes, but if the roof is nearly choked, the rock will not tilt in falling and so will not occupy any more space than it did before. Consequently the material that falls when the roof is nearly choked will not occupy any greater space than before falling and will, therefore, not give the roof that is unbroken any support. For this reason, I do not think any body of rock can fall till it chokes.

We talk about subsidence as extending beyond the pillar. That is usually a rather loose use of terms. It is true that there may be a trifling subsidence beyond the edge of the pillar; cracks are not the subsidence itself but the result of it. The measures over the pillar may be lifted by teetering on the edge of the unexcavated material, but this upward teetering may not raise the surface, though measurements have shown that it has done so in some instances. This failure to rise is due to the fact that the edge of the pillar may crush as the result of being used as a fulcrum and may sink into the bottom if the pillar rests on clay, thus lowering the fulcrum itself. The clay under a pillar is often extruded thus allowing

the edge of the pillar to descend. In this way, the measures though teetering upward, may go down with the fulcrum; and may lower rather than rise at the crack that has been denominated in this discussion as the "point," or rather "line of draw."

C. M. Young (written discussion).—R. D. Hall believes that so-called choking of a fall cannot occur. It seems to me that what is practically choking can occur. Let us assume a body of rock made up of thin layers, or easily separating into such layers, such as shale, and farther assume that no layer can fall until the layer immediately beneath it has moved so as to remove support. These assumptions are reasonable. If an excavation is made below such a layer, the immediate roof will fall and the rock will peel off layer by layer. The fallen rock will occupy more space than that in place and eventually the pile of debris will reach the top and support the layer still in place, which in turn supports that above it. It is true that such a pile may gradually become compacted, allowing more rock to fall, but this process may take so long a time as to escape observation. If above this friable shale there is a tough bed, some subsidence of the surface may occur eventually, but it is likely to be very small because of the support supplied by the pile of debris.

H. G. Moulton.—The effects of subsidence may extend outside the area involved and the subsidence itself may not extend as far as the effects of it do. Sometimes there is an actual extension of subsidence and sometimes only a crack resulting from it. In copper mining, extensive subsidence outside of the areas of excavation is very common.

G. S. Rice.—Probably it is right not to call that subsidence which is merely a lateral movement beyond the line of mine excavation, but it is an exceptional condition where tension is manifested without subsidence. Usually, where there is a deep central depression, a slide movement is created around and this may produce tilting of massive blocks.

R. D. Hall.—In the anthracite region of the United States observers have noted a rising of the surface over the pillar as a result of excavation.

G. S. Rice.—The character of the strata is very important, also the time element. If a movement of the surface does not occur for several hundred years, perhaps we can almost disregard it. But as regards support of roof or hanging wall from the loose material breaking off, if the area is large, say 10 acres or more, ultimately this ground will compact and allow settlement, which may take many years to affect the surface. When the excavation is narrow and deep below the surface, after the roof breaks and as the falls continue on to a central pile, the sides are gradually supported and the span lessened.

In an iron mine, where only part of the excavation had been back-filled, the fall has gradually gone upwards about 300 or 400 ft., but

has not broken through to the overlying sands. A tendency of convergence showed at the top of the arch of the natural stope indicating that this ground was breaking off, sliding off the pile of loose material and resting against the haunches of the arch.

R. D. Hall.—If the roof is to be supported by choking it must be by a long series of falls; even then the roof that has failed cannot support the roof above. After successive portions of the roof, however, have fallen till the space between the fall and the roof is quite small, the roof may sag without breaking until it rests on the fall. There is a molecular flow in all rock supported at its extremities permitting it to sag with time even without the imposition of any new load. The old-fashioned horizontal tombstone supported at its ends by stones and unsupported over the area above the grave has been observed to sag with the passing of decades and centuries. But the support of the roof of the mine in the manner suggested is obtained not by choking alone but choking with subsequent sagging and such an action could be expected to take many years or even centuries.

A. Locke, San Francisco, Calif.—We have been studying some subsidence in copper and gold mines which involves very much larger movement than any of the examples just cited. The two cases that have been accurately measured show a subsidence of 200 and 350 ft. respectively, at depths between 5,000 and 10,000 ft. The cause was the removal of many millions tons of rock during the primary mineralization.

These cases have some extraordinary characteristics. The walls are essentially vertical. In the Pilares mine in northern Mexico, the walls are so nearly vertical that one level looks just like over a depth of 1800 ft.; that is, the levels are all contained within an oval strip 100 ft. wide. We have essentially vertical or nearly vertical walls of oval or circular outline, localizing mineralization. They were hard to explain until we found that they were results of subsidence. Does any established set of conditions favor vertical walls?

H. G. Moulton.—My opinion based largely on studies of subsidence over copper-mining operations and subsidence in earth, is: The true behavior of materials failing under the stresses resulting from their own weight does not appear until the depths are great enough to result in pressures in excess of the sheering strength of the material; and when those depths are obtained, the probabilities are that the failure will extend outside of the vertical planes, and that the areas subsiding will be materially in excess of the area of extraction causing the subsidence. Generally speaking, a failure resulting from stresses due to weight of material in excess of its sheering strength of the material has a tendency to extend at an angle of approximately 70° from the horizontal. The line of break tends to be more or less parabolic, being straighter at the top and curved inward at the bottom.

Assuming that the material is homogeneous, an increase in depth tends to throw subsidence outside the vertical planes because it develops stresses that cause the material to fail on a grand scale, as distinct from smaller failures due to local planes of weakness; and time, which permits stresses to operate normally and without the complication of local stresses over brief periods at particular points will also tend to move the angle of draw or breakage further outside the vertical plane. Perhaps the prevalence of curved faces on very high cliffs, such as El Capitan in the Yosemite Valley, may have some bearing on the theory underlying the formation of surfaces of stability.

J. F. Kemp, New York, N. Y.—El Capitan is probably due to the fact that it is in a glacial valley with a "U"-shaped cross-section. Of course, the contour depends also on the joints of the rock and on the way in which the cracks develop when the rock breaks off, and what sort of surface they leave behind. Only the very soft rocks would be inclined to flow and mold together at the depths we reach in mining. The more resistant ones, from which the ores are almost always extracted, hold up quite well usually and when they yield it is in cracking blocks. If we were to try in shales at great depth we would reach a point when there would not be cracking but flowage, as we might expect dough or plastic clay to behave.

In 1912, I was greatly interested in observing the subsidence, both threatened and actual, along the prism of the Panama Canal where the grounds of the banks sometimes collapsed along and between fault lines, which were planes of weakness. At some places where the banks were especially high, as at Culebra, the working force were surprised to find the bottom, in the morning, higher than they had left it the night before. Overloading the sides caused the bottom to bulge by slow flowage of shales. The high banks were therefore reduced to ease the pressure.

J. B. Porter, Montreal, Can.—In the quotation from an English book that subsidence does not take place more than ten to one, was not a decimal point misplaced? Callon and Fayol who probably made the first study of this subject in France 60 years ago, said 100 to 1, and they have been quoted very freely. Fayol dealt with the arch on the supposition that the rock is sufficiently small and semiplastic to flow as it falls. On the other hand, if the rock is a slippery or shaly type, there is no doubt that it comes in from outside.

As to the effect of blasting shock, I never studied the problem from a level lower than where the blasting took place, but some years ago I made an extensive study of the effect of blasting upon the overlying strata. Using seismographs, under various conditions, I found that the disruptive effect was very much greater in clay rock or in clay strata than in any other kind. For instance, the shock of blasting in hard rock is very

sharp but the actual movement is very minute in distance and usually does not break the rock, whereas in plastic, or more or less plastic strata, within the reach of the blasting shock the ground is actually distorted and moves appreciably. The wave from the blasting, is virtually an earthquake wave and in such ground has quite a different length and period from that which it would have in rock, and displace the clay far more than it would displace rock. That may have some possible bearing upon the case that was referred to in Pennsylvania. It may have been that there was a layer of clay which would have a different wave period from the ordinary rock; it may have started a side movement which would, of course, cause the destruction of the pillars and then let the lower mine down not so much through actual breakage as through the initiation of creep.

Japanese studies of earthquakes have brought out forcefully the question of the shape of the wave, due to an earthquake shock. Blasting, of course, is precisely the same effect.

H. N. Eavenson.—If the decimal point in that English reference was misplaced, it would have been different in other statements. In our search through English literature, we saw it thirty or forty times, particularly in connection with the investigation of the Royal Commission over there, and they are very rigid about the amount of lateral support they require for public works.

The choking of the area has a great deal to do with the effect on an overlying seam within a short distance. It seems inconceivable to me that if the strata fell altogether, the overlying seams would not be very much disturbed. It is very unusual to find that at all. On the other hand, in cases where for depths of from 500 to 800 ft. the strata were from 60 to 70 per cent. very heavy sandstone ledges, we find heavy cracks and subsidences of from 1 to 2 ft. with 6 or 7-ft. thick coal-mined underneath. The choking effect must have been very slight or the measures settled and compressed the falls enough to show the subsidence.

There are cases mentioned in England, too, and any number of levels shown where the strata did actually rise in connection with longwall workings and then subsided. I saw five or six such instances.

H. G. Moulton.—Several examples of the subsidence of surface for causes other than mineral extraction are known. It is questioned whether or not parts of the Gulf Coast are subsiding as the result of extraction of oil. The Geological Survey made some study of this, and the geologists of some oil companies in Texas have it under consideration.

J. F. Kemp.—At Goose Creek, the wells were originally located on a low-lying but still dry shore along the shallow waters of a bay. As the wells proved very productive, the derricks multiplied and the outlines of the oil pool became quite sharply defined. The oil is in lenses of

incoherent sand, and the oil brought up much loose sand with it. As the oil and attendant sand were extracted, subsidence took place and the Gulf waters spread in around the derricks, in shallows of salt water. The subsidence was greatest over the central or maximum production and waned to nothing in the run of production.

Subsidence of oil territory following production in solid strata is practically unknown, and this experience led to the claim that the Goose Creek invasion of the water was due to the general subsidence of the Coast. In that event, it was claimed that the leaseholds reverted to the state and could be relocated. In the trial no general subsidence could be shown and it was therefore decided that the local sinking of the surface was due to removal of support in a section of unconsolidated and soft beds.

W. T. Thom, Jr., Washington, D. C.—The Goose Creek incident should cause us to proceed with caution in fields producing from unconsolidated oil sands. Subsidence that is not visible on the surface in other fields may have ruptured the originally impervious strata overlying the oil sands, so that further recovery by the introduction of gas or air or water may not be possible, at least not so readily as when we were dealing with a tight container.

One might regard overburden as being of three general types; (1) the homogeneous, massive type; (2) the interlaminated, alternating hard and soft beds; (3) the plastic type, which will yield as the chairman has suggested.

In the Colorado Canyon, in Utah and Arizona, where the rocks are largely massive, uniform sandstone, and where general uplift has taken place, the effect of this uplifting induced practically vertical fissuring, vertical lines of weakness. Although the rock is relatively weak, nearly all of the cliffs stand more than 1000 ft. high with practically vertical faces. So that in a region of general uplift, where one deals with massive beds, whether massive clays or massive sandstones, the vertical type of subsidence controlled by jointing will tend to develop, whereas the effect would be totally different were the overburden to consist of relatively plastic clays or were it composed of interlaminated hard and soft beds which would yield much as do beams.

H. G. Moulton.—On shoring work in the vicinity of New York—subway construction and excavation for buildings—quicksand seems to behave almost the same as hardpan, as far as the angles of break are concerned. There is a general law that governs the behavior of materials other than true fluids at the point where the pressures resulting from depth are in excess of shearing strength of the material.

Henry Louis, Newcastle-on-Tyne, Eng. (written discussion).—I disagree with the conclusion that subsidence resulting from bord-and-pillar

working covers a lesser area than the working itself or (as I prefer to put it) that in bord-and-pillar mining the draw is negative. I have never come across a case in this country that is in accord with that conclusion nor have I heard of any authenticated case where this is so. I am speaking of bord-and-pillar mining where the pillars are extracted and the area is goafed. Furthermore, I want to emphasize that I am speaking of subsidence of the surface as determined by careful leveling. Damage or even surface cracks often appear within the excavated area when accurate leveling shows that the subsidence extends beyond it. If my tentative theory on the cause of draw[8] is even approximately sound, the draw ought to be similar in direction, regardless of whether support is withdrawn in one operation (longwall) or in two (bord-and-pillar). I admit that in the latter case, the roof is likely to be more broken up and the total subsidence may sometimes be less, especially where the overlying strata are hard, but in neither case can I see any grounds for expecting negative draw. Of course, my remarks apply to horizontal or approximately horizontal seams.

I do not understand why the effect of bord-and-pillar mining on the surface should be so different in America from what it is in Europe and can only suppose that American engineers have not been taking levels but have been guided by the appearance of surface damage.

There is no place where the subsidence due to extraction of pillars can be better studied than in the Cleveland ironstone district. Here the workings are confined to a single bed of ironstone 9 to 10 ft. thick, lying relatively flat with dips, say, from 1½ to 3 in. per yd., and overlaid by quite regular strata of shale and sandstone of Mesozoic age and in places by some bowlder clay. Over a large area the seam is 200 to 500 ft. below the surface, which is often flat or undulating. The district is traversed by numerous railroads and main roadways so that accurate leveling has been possible. A number of different companies are at work, hence there are a large number of working centers.

The usual plan is to work the ironstone in pillars, usually 25 to 50 yd. long by 15 to 25 yd. wide and, after the narrow work in an area has been completed, the pillars are systematically worked out. While each pillar is being worked, the roof generally is carried on timber which is then drawn; the roof falls almost immediately forming goaf. In some cases, goafing follows close upon the narrow work; in others, the narrow work is completed in a fairly large panel before goafing commences. I have been unable to detect any difference as regards subsidence between these two methods.

I have had considerable experience in the effect produced on the surface by these mining operations. Immediately above the goaf, the surface

[8] Henry Louis: A Contribution to the Theory of Subsidences. *Trans.* Inst. Min. Eng. (1922) **64**, 257.

usually settles down 3 to 5 ft., the subsidence tapering off to zero around the affected area. In every case without exception the area thus affected is greater than the area goafed; in other words, the draw is always positive, which as a result of my experience, I should place about 60° on the average. It is probably less, say 56° to 70° in the harder sandstone and 50° in the bowlder clay (angles measured to the horizontal).

In this connection I would refer you to the evidence given by A. M. Hedley before the Royal Commission on Mining Subsidence, in which he gives the draw in sandstone beds as 68°, in normal beds as 63½°, and in bowlder clay as 45°. He never even considers the possibility of a negative draw. You will bear in mind that these angles refer to secondary rocks where the angle of draw is generally flatter than it is in coal measures. When T. A. O'Donahue was giving evidence before the same Commission he said that the area of subsidence in all cases is much larger than the area of working, and apparently he holds this to be equally true whether the working is longwall or bord-and-pillar.

The illustrations in the Committee's report seem to bear out my views. In most cases I find that reliance has been placed on surface cracks alone and these, as I should have expected, are within the area goafed and often are on the margin of that area. The last place, in my experience, is very usual. In a few cases I note that cracks are shown even over the solid coal; this I have also seen, but its occurrence depends on a number of circumstances. Where actual surveys have been made, as in Fig. 25 (plan 5–120) there is distinct evidence of subsidence outside the goafed area; Fig. 12 (plan 5–124) shows exactly the kind of surface action to which I am accustomed.

I do not believe in a negative draw. When a shaft top or an important structure, for example a water reservoir, must be protected against the effect of subsidence, a pillar of coal must be left in the underlying seam or seams. Our invariable practice is to leave a pillar of larger area than the area on the surface to be supported. If there are several underlying seams, the pillars left in the lower seams are always larger than those of the seam next above. If the draw were negative, in the case of bord-and-pillar workings, the supporting pillar of coal, if required at all, would be smaller in area than the area on the surface it had to support, and the pillar to be left in the lower seam would be smaller than that in the seam above. You will not find one responsible mining engineer in the United States who would set out his protecting pillars on that principle irrespective of whether the coal is to be worked by longwall or by bord-and-pillar.

I am far from wishing to dogmatize on so recondite and so complex a subject as subsidence and one concerning which our ignorance is so inexusably profound. Furthermore, you will understand that all I have written refers mainly to normal conditions with relatively flat seams and

where the country is not broken by faults or affected by dykes or other disturbances that may readily prevent subsidence taking place in a normal manner. I greatly appreciate the work that has been done, and sincerely hope that you will continue your efforts. I have contended for some time that the only way of obtaining facts is to select an unwrought area that is to be mined, to establish over it a series of monuments, and to make systematic levelings of these, say every 3 months, and at the same time to note carefully the position and progress of the goaf in the underground workings. If this were done in a number of places, we should have definite data on which to rely.

H. N. Eavenson.—The conclusions in the report are tentative and I agree with Mr. Louis that I cannot understand why the roof action over room-and-pillar mining should differ from longwall work, unless it is due to the causes mentioned. It is true that most of our cases deal only with cracks and subsidence that have developed and that have been mainly, but not entirely, within a worked-out area. There are some cases, however, where leveling was done and, unless some special features were involved to which attention was called in the report, the levels showed practically the same results as the evidences of subsidence did.

This question has never created as much interest in this country as in England, very largely because most of our mines are either under farming country or in heavily wooded mountains where the effects of subsidence are not of great importance and are usually negligible. Only on the outskirts of small towns or where pillars have been extracted around reservations protecting buildings has accurate information been obtained.

In Fig. 25 (plan 5–120) the mine area was not robbed but was left with the pillars in, as is shown on the plan, with the result that the area squeezed and subsidence resulted. Our experience in this country has been that in conditions of this kind the action is different from that of actual subsidence. In Fig. 12 (plan 5–124), the levels taken in the street over the worked-out area do not altogether show the action Mr. Louis mentions.

I partly agree with him regarding the negative draw. I cannot entirely believe that the draw is negative, although our experience so far indicates that this is the case. We are following one case which has been watched for 3 years, where the levels are being carefully taken. So far we have not been able to detect any subsidence outside of the worked-out area. The cover there is comparatively shallow. I do not think the mining results there are as yet complete.

In this connection, the cases cited on pages 32 to 38 of Mr. Louis' article in "Colliery Guardian" for the Indian Collieries apparently do not show any subsidence outside of the worked-out areas, and apparently levels were taken over enough area to indicate this. The same condi-

tion, I think, is largely true of the instance in Natal mentioned in the same article. The argument Mr. Louis makes about the shaft pillars is strong, but our methods in this country have largely followed those abroad, so that our practice does not necessarily represent the present opinions of the engineers here.

S. A. Taylor.—I would just like to give a word of caution on taking the breakage lines as an absolute line, as shown on these sketches by Mr. Eavenson. That final result of the break back of the solid coal is very close and is accurate in the cases given. I have seen, however, instances where that might not be the case, and the same thing that I pointed out a little while ago regarding the character of the overlying strata has much to do with it. In other words, where the extraction takes place the breakage is not in a straight line back to the point shown on the surface, but probably comes out over the front or excavated part of the coal and then goes back over the top of the coal, so that if you were to take a line or an angle there of 2° or 10° or 18°, you might get into difficulties.

H. G. Moulton.—One of the most indeterminate features today in connection with mine subsidence or breakage of ground is the question of the shape of the plane of fracture between the surface and the line of failure in the underground workings. At one time a shaft was sunk in New York City in an attempt to follow the line of break in earth. Excavation at a given point had caused the break and some 45 ft. outside the vertical plane of this point a crack had appeared. A shaft was sunk on the crack for about a third of its depth until it was lost in boulders and sand. In that case the crack extended vertically into the ground, yet somewhere below it must have swung over and connected with the underground break that caused it.

The connection of the surface break with the underground failure by a straight line is clearly incorrect, but practically no data is available by which the line of the irregular fracture can be predicted. We hope that all who have an opportunity to observe mining operations will search in intermediate levels for breaks resulting from work carried on below and extending to known points above, for the Committee is particularly anxious to receive any data which will be of assistance in tracing the line of fracture below the surface.

S. A. Taylor.—In one case we had a reservation under a house on a hillside and we worked out to the reservation or within a few feet of the reserved line, and the break was very distinctly inside. You could see it coming down toward the hole. Yet there was a break following clear under the surface and it ruined the house that was on the solid coal. It was shown very clearly down underneath that the breakage was back for a considerable distance, probably 20 or 25 ft. back of the line, before

it seemed to take the break in the reverse way and brought down the house.

R. D. Hall.—The suggestion has been made in England that the forward break goes to the neutral axis of the main roof (that is, the real roof not including the draw slate) and that from that point it goes back to the point or line of draw on the surface. The theory, however, is largely supposition.

G. S. Rice.—That would be modified by the character of the strata probably.

R. D. Hall.—It is assumed that the strata constitute a monolith.

H. G. Moulton.—Also modified by the fact that an irregular mass of rock probably has no true neutral axis.

R. D. Hall.—I am disposed to believe that in most cases it has. (The theory propounded in England has the disadvantage that the neutral axis is not a fixed line or rather plane but moves when the rock begins to break, and if the rock breaks first from the bottom up it would continually move upward till at length it would be near the surface.)

Subsidence around a Salt Well

C. M. YOUNG,* LAWRENCE, KANS.

(New York Meeting, February, 1926)

WHEREVER salt is extracted from the ground as an artificial brine produced by pumping down fresh water to dissolve the salt, subsidence of the overburden is a possibility, though apparently few cases of such subsidence have been noted and detailed records are wanting. This description is intended as a record of subsidence that may permit the forcasting of similar occurrences in other cases; the data were supplied largely by R. B. Lee, city engineer of Hutchinson, Kans. About eleven years ago, ground movement occurred at a neighboring salt plant, when the surface assumed the form of a shallow funnel. Movement was probably due to a flow of sand into a well through a poor seal or a broken casing.

The salt deposit increases in thickness from the east until at Hutchinson, where the subsidence occurred, it is about 310 ft. thick. The interbedded layers of shale vary in thickness from mere traces to about 7 ft. While knowledge of the shale content of the beds is not exact, the log of a shaft 3 miles east of Hutchinson indicates that shale forms about 25 per cent. of the total thickness. This shaft does not reach the bottom of the salt by about 60 ft., but there is no reason to think that the lower part of the bed is different from the upper. Logs of wells sunk to the bottom are not sufficiently detailed to be of much value.

Above the salt is the shale of the well-known Red Beds. The thickness of this is variable, principally because of its eroded surface, but at the mine it is 340 ft. This shale is variable in character and strength, but contains no strong members of limestone or sandstone. If the support is removed, very probably it will subside. The Arkansas River flows across the salt district and Hutchinson is situated on the flood plain. Between the top soil, which is about 8 ft. thick, and the shale is 60 to 100 ft. of water-bearing sand and gravel.

The process of removing salt, at least in the case of the earlier wells with which we are concerned, is as follows: An 8-in. hole is put down through the sand and gravel by means of a drive pipe, which is bedded in the shale. From the top of the shale, a 6-in. hole is drilled through the shale, to, or nearly to, the bottom of the salt. Casing is put in to a depth of about 140 ft. to shut off a water-bearing layer in the shale.

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A smaller pipe is then put down inside the casing nearly to the bottom of the well. Through this fresh water is pumped, which dissolves salt and ascends through the well as nearly saturated brine. In the case under consideration five wells were in use, the locations being shown on Fig. 1.

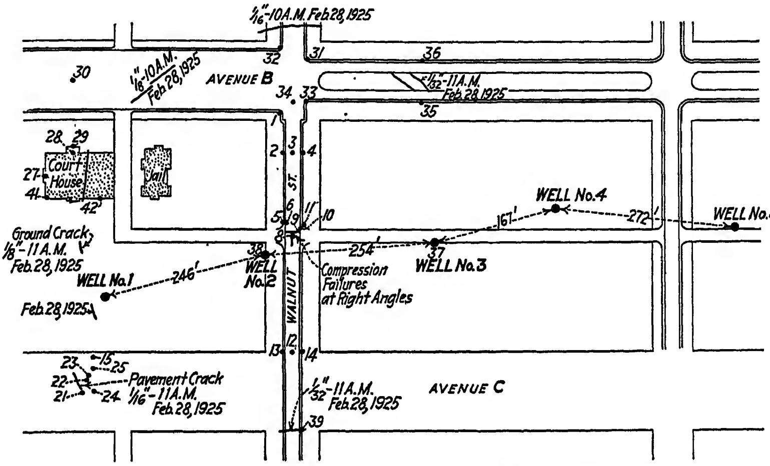

FIG. 1.—LOCATION OF WELLS, SURFACE CRACKS, AND COMPRESSION FAILURES.

SUBSIDENCE

The first sign of subsidence was the breaking of a supply pipe to well 2 during the night of Feb. 24, 1924, or the next morning. It is not known whether this was related to the subsidence; it may have been a coincidence. The first phenomenon definitely connected with ground movement was the sticking of doors in the court house on the 27th. No cracks had then been noticed but they appeared soon afterwards.

As soon as the evidences of subsidence became noticeable, the city and county engineers began a series of observations. They found a series of cracks roughly outlining a circle about 600 ft. in diameter with the center near well 2. About 50 ft. northeast of the well, near a manhole 8, the pavement buckled in two lines—east-west and north-south. Observations of the fluid level in the sewer indicated a vertical movement of about 6 in. City and county officials became alarmed, the county offices were removed to temporary quarters, and Dr. R. C. Moore, State Geologist, was called in for consultation. An attempt was then made to ascertain conditions at well 2.

An exploring rod about 13 ft. long was lowered into the well by two wires, one attached at each end. The purpose was to lower the rod

vertically through the inner tubing and then, by pulling the rod into a horizontal position, determine the upper limit of the chamber. The conclusion reached was that the tubing was broken off at a depth of about 290 ft. and that the top of the debris pile was at about 365 ft., only a little below the bottom of the shale. A test was made for the presence of sand, with negative results, but probably this should be considered inconclusive. The presence of sand might have indicated that the subsidence was due to the flow of sand into the well, but the nature of the movement indicates a subsidence of the whole overburden.

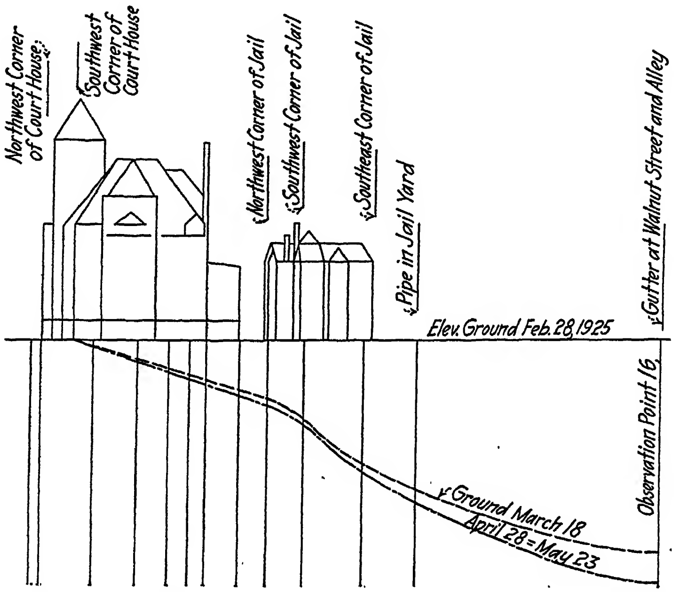

FIG. 2.—COURT HOUSE AND JAIL PROJECTED ON RADIUS FROM OBSERVATION POINT 16.

During this movement, well 2 was flowing; and even in May—after movement had ceased or become imperceptible it showed a pressure of 4.5 lb. The wells have been sealed and abandoned.

The form of subsidence and relation to the wells is shown in Fig. 1; the numbers show the observation points of the city engineer. The only building plainly showing cracks is the court house, which stands upon the line of surface cracks showing tension. The county engineer reports horizontal movement of the east end away from the undisturbed west end of approximately 2½ in. The county jail, which stands wholly within the area of movement, shows no cracks.

The extent of vertical movement after the beginning of observation is shown in Fig. 2, which was prepared by the county engineer. This is not total movement but only observed movement after the beginning of

subsidence manifested by sticking of doors in the court house. Progress of the subsidence, after the beginning of observations, is shown in Table 1, which was prepared by the city engineer. Numbers refer to observa-

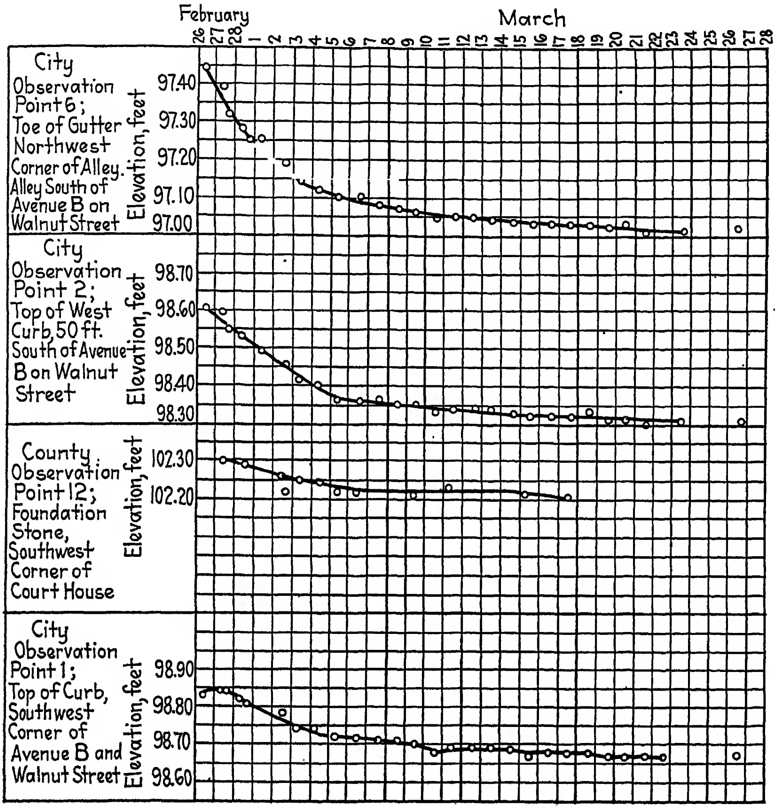

FIG. 3.—DEGREE OF SUBSIDENCE AT DIFFERENT OBSERVATION POINTS.

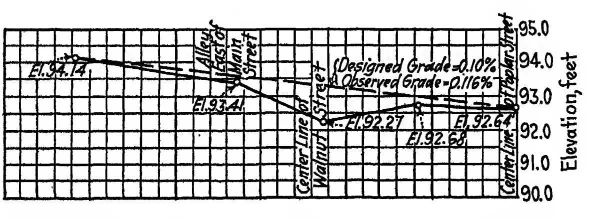

FIG. 4.—PROFILE OF 8-IN. SANITARY SEWER IN ALLEY BETWEEN AVENUE B AND AVENUE C.

tion stations. Forty-six stations were observed, but only a few clearly showing movement are reported here. The total motion is best revealed by Fig. 4 which shows the vertical movement of the sanitary sewer in the

alley between Avenue B and Avenue C. This indicates a subsidence of about 1 ft. at the point of greatest vertical movement.

TABLE 1.—*Progress of Subsidence*

Time of Observation		Observation Point				
Day	Hour	6. Toe of Gutter Northwest Corner of Alley	2. Top of West Curb on Walnut St., 50 ft. South of Ave. B	12. Top of Pavement, Center Line of Walnut St. at Ave. C	1. Top of Curb, Southwest Corner Ave. B and Walnut St.	32. Top of Curb, Northwest Corner Ave. B and Walnut St.
February 27..	11:00 a.m.	99.44	98.60	97.85	98.83	
28..	4:00 a.m.	97.39	98.59	97.83	98.84	
28..	4:15 p.m.	97.32	98.55	97.85	98.84	
March 1....	9:45 a.m.	97.28	98.53	97.85	98.82	
1....	6:00 p.m.	97.25			98.80	
2....	8:30 a.m.	97.25	98.49	97.82	98.80	99.01
3....	3:30 p.m.	97.19	98.45	97.82	98.78	
4....	9:00 a.m.	97.14	98.41	97.78	98.74	98.99
5....	8:30 a.m.	97.12	98.40	97.79	98.74	98.98
6....	9:30 a.m.	97.10	98.36	97.79	98.72	98.98
7....	2:30 p.m.	97.10	98.36	97.79	98.72	98.98
8....	3:00 p.m.	97.08	98.36	97.78	98.71	98.98
9....	9:30 p.m.	97.08	98.35	97.78	98.71	98.98
10....	2:00 p.m.	97.06	98.35	97.78	98.70	98.98
11....	3:30 p.m.	97.05	98.35	97.76	98.68	98.96
12....	4:00 p.m.	97.05	98.35		98.69	98.98
13....	3:00 p.m.	97.05	98.34	97.77	98.69	98.97
14....	2:30 p.m.	97.04	98.34	97.79	98.69	98.97
15....	4:30 p.m.	97.04	98.33	97.76	98.69	98.98
16....	4:00 p.m.	97.03	98.32	97.76	98.67	98.96
17....	3:30 p.m.	97.03	98.32	97.76	98.68	98.97
18....	2:30 p.m.	97.03	98.32	97.76	98.68	98.97
19....		97.03	98.32	97.77	98.68	98.97
20....	3:30 p.m.	97.02	98.31	97.75	98.67	
21....	5:00 p.m.	97.03	98.31	97.76	98.67	98.96
22....	3:30 p.m.	97.01	98.30	97.75	98.67	98.96
24....	4:00 p.m.	97.01	98.31	97.75	98.67	98.96
27....	5:00 p.m.	97.02	98.31	97.76	98.67	98.97
April 2....	11:00 a.m.	97.01	98.31	97.76	98.67	98.97
May 22....	8:00 a.m.	96.98	98.28		98.65	

Table 1 shows that movement had practically ceased by March 23. The decrease in rate of movement as a condition of stability is approached indicates support of the overburden on a cushion of broken rock, which slowly yielded as the load came upon it, at the same time increasing in resistance until it now sustains the weight without farther yielding;

this is well illustrated by the progress curves, Fig. 3 drawn from the data in Table 1. Fig. 5 shows a series of observations north-south across the disturbed area, crossing the middle of the line shown in Fig. 4. Subsidence and buckling are indicated.

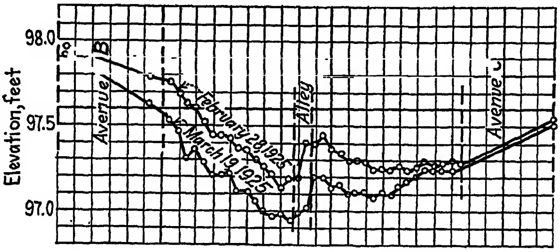

FIG. 5.—PROFILE OF FLOW LINE OF WEST GUTTER ON WALNUT STREET, SHOWING COMPRESSION.

UNDERGROUND CONDITIONS

The history of conditions underground from the beginning of solution of salt to the subsidence apparently resulting is an interesting field for conjecture. We have no means of obtaining knowledge but events have supplied indications. The most important inquiry concerns the shape and dimensions of the underground cavity, for if these can be known we shall have a foundation for prediction in other cases.

Water is pumped to the bottom of each well to dissolve salt and return to the surface as a saturated or nearly saturated brine. In the course of operations the tubing sometimes breaks off, generally near the bottom at first. Apparently this is because blocks of salt or shale carry it away in falling. If breaks occur too near the upper limit of the salt, pumping must be slow or brine of low salt content must be used, either condition necessitating abandonment of the well.

As to the size of the cavity, a rough approximation is possible. The salt company states that about 2,500,000 bbl. of salt have been produced from the five wells. A barrel of salt is 280 lb. and the specific gravity is 2.1, giving a volume in the ground of 2.133 cu. ft. per bbl. or a total of 1,172,000 cu. ft. Though something more than one-fifth of this should be assigned to the earlier wells because of their longer operation, it is impossible to be exact and the average excavation may be taken as about 235,000 cu. ft. If a cylindrical chamber were excavated through the thickness of the salt it would be only 758 sq. ft. in area and the diameter would be 31 ft. However, it is known that wells 1, 2, and 3 are connected underground. There is no reason to assume that solubility is not the same in all directions; therefore we must suppose a circular chamber around each well with a radius of about 125 ft. The

shape of this chamber is unknown. Salt is soluble most readily in fresh water and we might expect a conical or pear-shaped cavity. It seems probable, however, that the shape is modified by shale beds. These are insoluble and the flow of water is so slow that erosion may be neglected. They would therefore protect overlying salt just as resistant strata on the surface protect weaker underlying beds and form plateaus bordered by escarpments. The result would be a low, broad opening. Eventually the overlying salt would fall and there would be a chamber nearly filled with broken salt and shale. From the profile shown in Fig. 2, it appears that there has been little arching, but the salt seems to have fallen over an area nearly equal to the greatest extent of the chamber. When the supporting salt was removed, the shale settled down bodily as it does in longwall mining, with the resultant draw extending beyond the lines where support failed.

If the chamber has walls nearly vertical only about 2 per cent. of the volume has been removed as brine and we may suppose that the chamber is nearly full of blocks of salt and shale, which act as a support for the overburden. The yielding and consolidation of this cushioning support have already been mentioned.

The principal conclusions to be drawn from this occurrence seem to be:

That the excavation by water is broad rather than high and narrow.

That the salt and shale will not arch but will break to form a cavity with nearly vertical walls.

That the overlying shale will subside bodily when the caving extends to the top of the salt.

That the vertical movement will be small because of support supplied by broken salt and shale which occupy most of the volume of the chamber.

DISCUSSION

H. G. Moulton, New York, N. Y.—There are a number of examples of subsidence over extended areas in mining operations as a result of the extraction of salt or sulfur through wells driven to considerable depths, but the Committee has been unable to secure much data with respect to relative measurements.

In such cases as the one described it is possible to determine both the vertical and the horizontal extent of the subsidence, and consequently both the area and the volume involved; it is also possible to study the progress of the movement by time intervals. However, it is difficult to obtain any figures from which we can determine the relation of the area of subsidence to the area of extraction.

When some of the salt wells now being operated in various parts of the country, and the sulfur properties extracting sulfur by the introduction of hot water, have worked out and definitely abandoned certain areas

so that the limits of underground extraction may be determined by interpolation between productive and non-productive wells, the Committee should be able to obtain some definite information along this line, and we shall appreciate any assistance that we can secure in obtaining the necessary data.

R. D. HALL, New York, N. Y.—In North Wales, brine has, for many years, been pumped from under the city of Chester. As a result houses and streets have repeatedly gone down and been invaded by the lakes consequent on the interference with drainage that such subsidence has created in the surrounding country. Sometimes there is a fracture to the surface, and then the water is thrown into the air to a considerable height. I do not doubt but what some instruction could be obtained from information drawn from this source, although that is not exactly the condition which we are desirous of studying.

W. T. THOM, JR., Washington, D. C.—The salt dome formerly known as Big Hill in Matagorda County, Texas, was formerly marked by a mound, which stood 35 ft. or more above sea level. During the extraction of sulfur by the Frasch process, a pond has been developed over at least a part of the site of the former hill, indicating a total subsidence of perhaps some 35 or 40 feet.

Factors Affecting Bank Slopes in Steam-shovel Operations

By Louis S. Cates*, Salt Lake City, Utah

(New York Meeting, February, 1925)

At the annual meeting of the American Institute of Mining and Metallurgical Engineers in February, 1923, the Chairman of the Committee on Ground Movement and Subsidence appointed a sub-committee to work out a form of questionnaire and to collect data on subsidence and ground movement resulting from underground metal and coal-mining operations, and on ground movement and safe slopes in connection with open-cut metal-mining operations. The author was requested to take charge of the collection of data and the preparation of a questionnaire in connection with the Committee's studies of open-cut metal-mining operations. This paper is a summary of the work carried on and information received therefrom up to February, 1924.

The first approach to the problem of safe slopes is to gather all information that pertains to finished slopes; in other words, the steepest economic angle from the horizontal that will stand in order to uncover the ores to be mined and be safe while the extraction of the ore at the base of these slopes is in progress. Whether or not this angle is 35° or 45° depends on many factors. The number of benches maintained to take care of falling rock and their height and width are direct functions of this overall slope. The slope is here taken to mean the angle from the top edge of the pit through the edges of the respective lower benches to the bottom of the pit. If we accept this definition of the slope, it might save some confusion as there would be a difference if one took the line from the top edge of excavation to the toe of the bottom slope. This latter slope would not be constant, but would vary with the number of benches.

The depth of the various ore faces will vary, of course, depending on the deposit; Table 1 shows this to range from 50 ft. to 1100 ft.

The geology is an important factor, as are the climatic conditions under which one is operating. A saturated condition in some rock structure causes rock flows; in the Panama Canal excavation, slides have run on a ratio of as low as one on ten, which is a 10 per cent. grade, or a slope of less than 6° with the horizontal. Slopes as steep as 60° have been noted in the Utah Copper operations, so that a wide divergence will possibly be met. The work of securing data that will be of value to the operator or estimating engineer will be difficult and will mean little unless a detailed

* Vice-president and general manager, Utah Copper Co.

TABLE 1.—*Open-cut Mining—Ore*

1	Name of mine	Ajo Open Pit, Ajo, Ariz.	Copper Flat, Ruth, Nev.	Sacramento Hill, Bisbee, Ariz.	Bingham Shovel Pit, Bingham Canyon, Utah
2	Name of company	New Cornelia Copper Co., Ajo, Ariz.	Nevada Consolidated Copper Co., McGill, Nev.	Phelps-Dodge Corpn., Copper-Queen Branch, Bisbee, Ariz.	Utah Copper Co., Kearns Bldg., Salt Lake City, Utah
3	Kind of ore	Copper	Porphyry copper	Sulfides in porphyry	Copper
4	Character of orebody	Disseminated oxide and sulfides in hard silicified monzonite porphyry	Sulfides in porphyry	Sulfides in porphyry (See original for detailed information)	Disseminated sulfides in porphyry
5	Thickness of orebody to be mined by open cut	Average, 250 ft.	500 ft.	Average, 227 ft.	556 ft.
6	Character of overburden	No overburden	Leached porphyry and altered limestone	Brecciated and silicified porphyry and altered limestone	Leached porphyry and quartzite
7	Thickness of overburden	As above	Maximum, 200 ft., average, 110 ft.	50 to 300 ft., average about 150 ft.	Average, 115 ft.
8	General method of mining	Steam-shovel benches	In benches with steam shovels	In benches with steam shovels	Steam and electric shovels in benches
9	Vertical height of entire face of ore	40–50 ft.	Maximum, 150 ft.	Maximum, 120 ft.	1100 ft.
10	General angle of slope of ore face	Not established	36°	Not determined	35°
11	Angle of slope of ore between benches	About 50°	45° to 60°		45°–60°
12	Vertical height between benches	30 ft.	30 to 90 ft., usually 50 ft.	30 to 60 ft.	54 to 222 ft., average 70 ft.
13	Angle of repose of broken ore	45°	32°	37½°	34°
14	Angle of repose of overburden	No overburden	32°	37½°	34°
15	Angle of slope of overburden in place	No overburden	55° to 60°	Slopes of 45° are left with 15-ft. benches. The slopes between benches are steeper than 45°.	50°–60°, working slopes in overburden
16	Width of bench between overburden and ore.	As above	30 to 50 ft.		50 to 70 ft., width working benches
17	When was mining started, date	May, 1907	1907	April, 1917	August, 1906
18	Dates of observations and surveys	Data taken monthly, date forwarded, June '23	June, 1923	From recent progress surveys	January, 1923
19	Accompany report by profiles (dated)	Section "L" through open-pit mine	Profiles and sections	Sections showing ore and pit outlines	Photostat sections
20	Photographs of slopes and benches	No photographs	No photographs	Photographs	Photographs
21	Signature of observer	John C. Greenway	W. R. Brown	Gerald Sherman	William Spencer

description of the conditions existing is given. Possibly no two operations will be alike in this regard.

The reason for the importance of this question is understood, when the difference in the yardage necessary to excavate is noted as the slope flattens. Whether or not low-grade ores are commercial, may depend on the amount of overburden to be moved to effect their recovery; and the amount of overburden will vary with the angle of repose of the solid bank.

If 60-ft. banks are carried with 30 ft. ultimate width of benches and 45° slopes assumed on the individual benches, the overall slope will be (as before, taking a slope angle through the edges) about 33½°, while if 120 ft. heights, 30 ft. widths, and 45° slopes are used, the overall slope will be approximately 38°. By the elimination of benches then, the slope steepens. From our experience at Utah, we believe that we can hold slightly over 50° slopes on the benches and, without eliminating any levels, hold an overall slope of approximately 38°, considering a final 30-ft. width. When it becomes possible to eliminate some of the benches and still accommodate the haulage, it may be possible to hold 42° or perhaps slightly more (see Fig. 1).

There is room for discussion on the final width of a bench. If the rails are to be maintained, with the idea of possibly operating the bench again, 30 ft. will probably be a minimum; though in the end this might be narrowed to one-half that amount, leaving a berm for safety only. A good illustration of the conditions prevailing during operation is shown (Figs. 2 and 3).

With conditions at Utah, where heights upwards of 1000 ft. are to be considered, we are perhaps more concerned with some of these problems than would be the case at shallower deposits; see Fig. 4.

Slopes involve property limits; and final subsidence after mining operation has ceased might cause damage that would no doubt affect some operations, so that while, at first, the discussion might seem a simple one, the final analysis is far reaching.

In order that the collection and correlation of data might be simplified and expedited, the questionnaire suggested by George S. Rice, Secretary of the Committee on Ground Movement and Subsidence, in his paper at the February, 1923, Meeting, has been sent to a number of mine operators and engineers, and replies have been received from the following:

Gerald Sherman, consulting engineer, Phelps-Dodge Corpn.

C. B. Lakenan, general manager, Nevada Consolidated Copper Co.

W. S. Larsh, general superintendent of mines, Nevada Consolidated Copper Co.

W. R. Brown, chief engineer, Nevada Consolidated Copper Co.

John C. Greenway, general manager, New Cornelia Copper Co.

Earl S. Hunner, manager, M. A. Hanna Co.

The information supplied is given in Table 1.

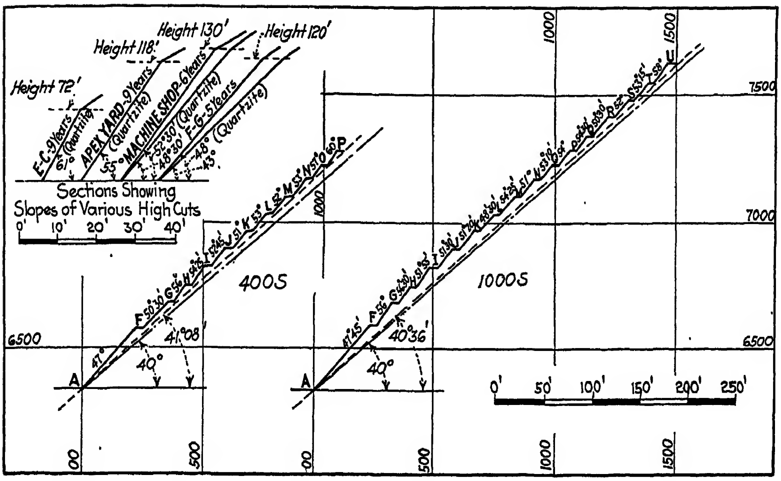

FIG. 1.—SECTIONS 400S AND 1000S, UTAH COPPER CO., SHOWING GENERAL SLOPE OBTAINED BY USING AVERAGE SLOPE OF BANKS FOR 10 YR., WITH 30-FT. BENCHES.

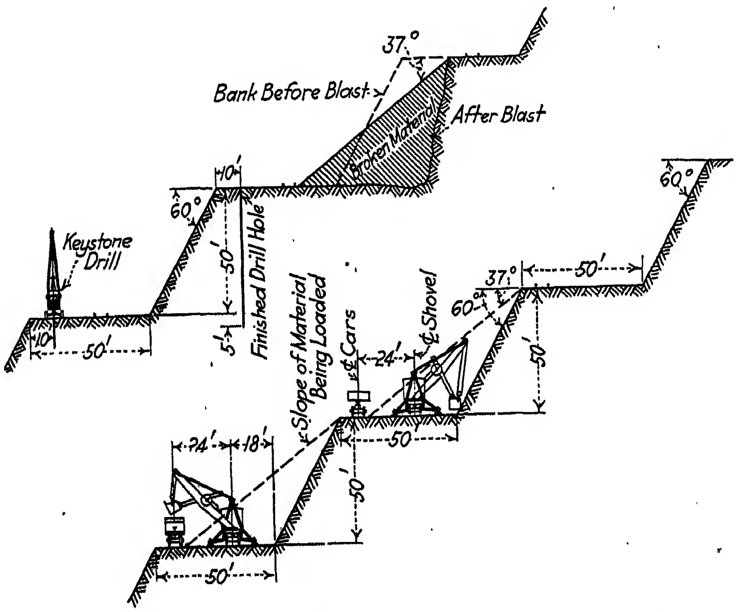

FIG. 2.—SECTIONS OF LEVELS OF NEVADA CONSOLIDATED COPPER CO., AT RUTH, NEV., SHOWING WORKING AND BLASTING CONDITIONS.

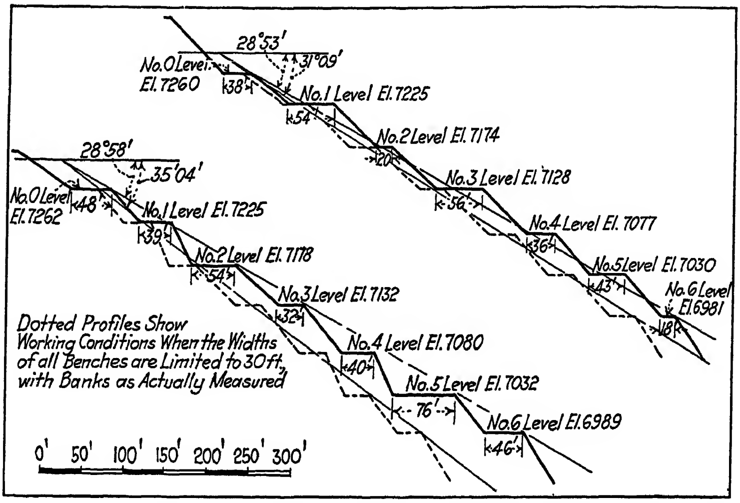

FIG. 3.—ACTUAL PROFILES OF WORKING LEVELS OF NEVADA CONSOLIDATED COPPER Co.

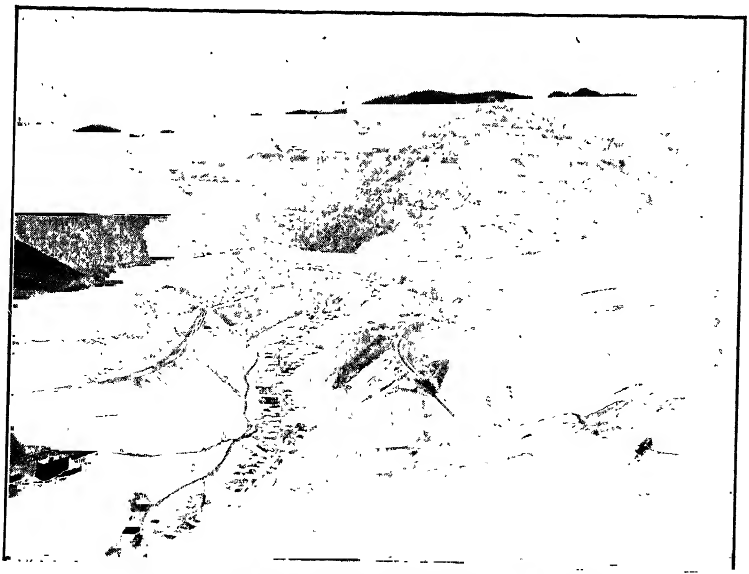

FIG. 4.—UTAH COPPER PIT AT BINGHAM, UTAH.

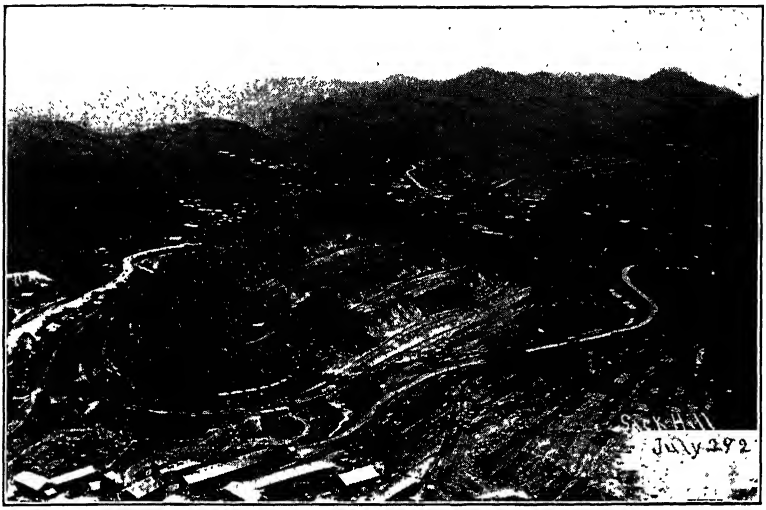

FIG. 5.—GENERAL VIEW OF SACRAMENTO HILL PIT, BISBEE, ARIZ.

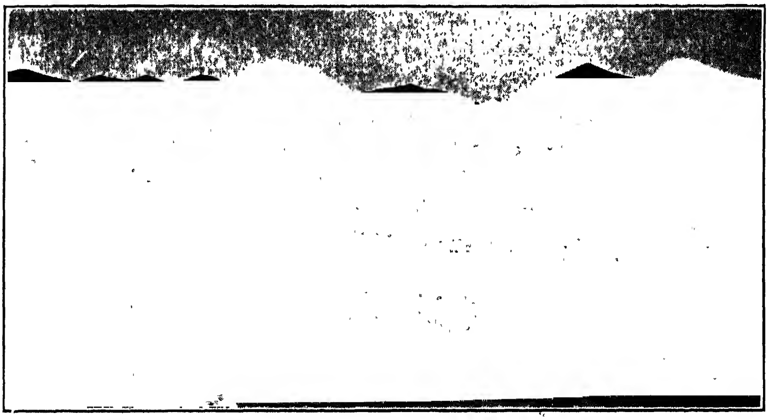

FIG. 6.—TYPICAL BENCHES IN SACRAMENTO HILL PIT.

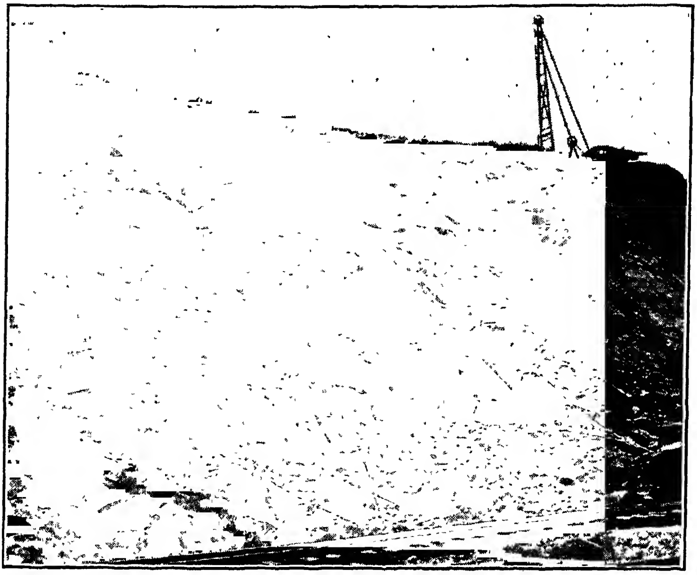

FIG. 7.—FRONT VIEW OF OPERATING BENCH, SACRAMENTO HILL PIT.

FIG. 8.—SIDE VIEW OF OPERATING BENCH, SACRAMENTO HILL PIT.

A number of photographs and vertical sections showing conditions obtaining at representative properties have also been received. In addition to the sections and photographic views already cited (Figs. 1 to 4 inclusive), there are included a general view (Fig. 5) and a closer view of several benches (Fig. 6) and detailed views of operating benches (Figs. 7 and 8) of the Sacramento Hill pit of the Phelps-Dodge Corpn. at Bisbee, Ariz.

It is evident that work at the properties studied has not progressed far enough to establish finished slopes. The conclusions thus far drawn are all that can be reached from the data thus far available. In general, estimates covering the future operating details of steam-shovel property are based on an assumption of 45° slopes, which is all that can be done until such time as the nature of the banks can be studied and the ultimate limit of safe slopes determined. It is understood, of course, that where further ore developments are probable, but if for any reason the area cannot be explored satisfactorily in advance, all the benches will be maintained for future use if necessary. From the studies thus far carried on by the Committee, it appears that the ultimate slope left when a steam-shovel pit is worked out may be quite different from that found most satisfactory for working conditions during operation.

The questionnaire which is being sent out, and will be sent to anyone interested in the subject on application to the author, is designed to develop all possible existing information and all available views on the subject of safe slopes. It is desirable to obtain full sections and typical photographs, similar to the illustrations given here.

The purpose of this preliminary report is to make public the information thus far obtained and to stimulate the collection of further data, so that the Committee may be able to publish an extensive report on the subject.

DISCUSSION

Fred Hellman, New York, N. Y.—My experience with shovel mining has been restricted almost entirely to Chuquicamata as far as operation is concerned. We had there an unusual condition in that the rock is cohesive because of the salts in it, so that you can probably work higher banks than is possible in most places. Recently they have reduced the height of the banks and have found that lower bank affords better working conditions than the higher bank. It took two years of experimentation to determine the best height of bank for that particular mine. Each mine should be considered by itself and to state offhand the best height of bank for any given mine is impossible.

High banks were adopted to obtain the minimum expense in railway approaches at that time. The first plan was for 50 ft. banks, but to get production started with the smallest possible expenditure of

money, the approach to the deposit was changed and the first bank had an ultimate height of about 180 ft.

They could not work such high benches in Panama because of the great amount of lubricating minerals, such as sericite, talc, etc., which cause slides to take place on very low angles. The engineer in charge of the Culebra cut showed me a slide on 11°. The coal seam in the formation seemed to be only about 1/4 in. thick, and some hundreds of thousands of tons had come down on that 11° slope.

The cohesiveness of the rock at Chuquicamata is shown by some work done possibly before the Spanish Invasion in 1545. A huge cavern had been worked down from the top, until it was possibly 100 ft. long by 80 ft. wide and 80 ft. high; spanning that whole opening was a vault of perhaps 12 in. thick that had stood throughout the years. Even our heavy blasting did not shake that down for some time.

With sloping banks and tunnel, blasting the tunnels say 10 or 15 ft. below the bench, you blow out through the line of least resistance which is supposed to bisect the angle. Then the top of the bank comes down without any shattering from the blast. The great pieces are sometimes 12 ft. cube; they are difficult to handle with a shovel, and require a great deal of blasting for their breaking.

The benches often consist of materials of greatly differing character; there would be soft spots and hard spots. By loading the hole up to within 30 ft. of the top with different kinds of explosives—that is weak or strong—it is possible to shatter this rock, and by putting high-grade explosives behind the hard spots, a shattering effect is produced behind them. In other places it is possible to break out and shatter the rock with a lesser explosive. This plan of differential loading of the holes has been entirely successful.

Louis S. Cates.—Many people wonder why we have both high and low banks. Our experience shows that the low bank is by far the better. But we have one bank (our first) 220 ft. high. Within the last year or so we have not been able to get the switches in the farther end of the quarry. Of course, the rock removed from the high bank is more expensive than from the low bank primarily, because "bulldozing" is required on the high bank and not on the low banks.

Fred Hellmann.—In Chile, we considered the question of the average slope very carefully for we were going down all the time, and could not decide how far we were going to drive back the upper benches. We had to supply some means of resuming work on these benches at some later date. A berm materially steadies the slope and is a good thing; it is unavoidable in some cases. It is much easier and cheaper to leave the berm as you go along. Most people working on high benches, such as are used in Utah, would I imagine leave these berms as a matter of safety.

W. D. B. Motter, Jr., New York.—We figured the ultimate slopes of the final pit at Chuquicamata at 60° between benches, based on the then existing bench height of 140 to 150 ft. and leaving 16-ft. berms on each bench. Since subdividing the benches we have not laid out a new system but we are going to see if it is practicable first to leave berms on each bench and then eventually to absorb the berms on the intermediate benches so that the final berms will exist at intervals of about 150 ft. The lack of rain and the cohesiveness of the rock change the factor so thoroughly that we may be able to achieve a much higher slope than 38°.

G. S. Rice, Washington, D. C.—In the case of these very deep cuts in open-cut work, is it not going to be necessary to take into consideration a certain critical depth, which varies with the class of material, dryness, weight, lateral pressure, etc.? For every class of material, there surely must be an ultimate loading that will cause a lateral movement at the foot akin to hydraulic effects. Many of you have seen cases of high rock dumps on soil and clay where the weight of the dump has caused the surface to be thrown up beyond the toe of the dump slope. Even for a strong material such as granite, you will get a pressure where the granite is going to break and flow, as indicated in the Lake Superior copper mines, and open-cut work would require the use of a flatter slope in the lower part of the cut. In a testing machine where a flat piece is crushed, the specimen will show radical lines of flow from the center.

Fred Hellman.—What is the total height from the bottom of a big bench to the top?

Louis S. Cates.—About 1700 ft., but I do not think we have reached the critical depth Mr. Rice mentions. We have been operating our mines on a 28° slope.

Problems of Pumping Deep Wells*

BY LESTER C. UREN,† BERKELEY, CALIF.

(Casper Meeting,‡ August, 1925)

WITH the depletion of our older, and relatively shallow, oilfields and the necessity for securing new production from deeper horizons, much attention is being given to the improvement of oil-well pumps in order that they may function satisfactorily under the more difficult conditions imposed by increased depth. Until recently, practically all of the world's petroleum was secured from wells less than 3000 ft. deep, whereas horizons are now being explored to depths of from 5000 to 7000 ft. in some of the more prolific fields, and it seems probable that the greater part of the future supply must come from depths in excess of 3000 ft. While much of the early production in a new field is obtained by natural-flow methods, when the stimulus of high gas pressure fails, recourse must be had to some method of pumping. The perfection of a pump capable of economically and efficiently lifting oil from depths ranging from 3000 to 7000 ft. is therefore a matter of prime importance to the oil-producing industry. This paper is offered as an estimate of the present state of development of the oil-well plunger pump and as a review of the general principles involved in its operation in deep-well pumping. A design for a new type of oil-well pump embodying several novel features, is also described.

ESSENTIAL PARTS OF OIL-WELL PLUNGER PUMP

The pump commonly used in lifting oil from wells in American practice is of the simple displacement type, in which a plunger equipped with a working, or traveling, valve is given a vertical reciprocating motion in a stationary working barrel. The latter is suspended from the surface by tubing, submerged in the well fluid, and is equipped at its lower end with a stationary standing valve; Figs. 1 and 2 illustrate two common types of oil-well pumps. The simple cup-leather packed plunger of Fig. 1 has its counterpart in the long, hollow, cylindrical, polished steel plunger of Fig. 2; otherwise the two types are identical in the form and arrangement of their essential parts. Fig. 2 shows the additional feature of a "garbutt

* Petroleum Development and Technology in 1925, 130. (A. I. M. E. Pamphlet No. 1570-G, issued with MINING AND METALLURGY, April, 1926.)

† Associate professor of petroleum engineering, University of California.

‡ Mid-year meeting of Petroleum Division, A. I. M. E.

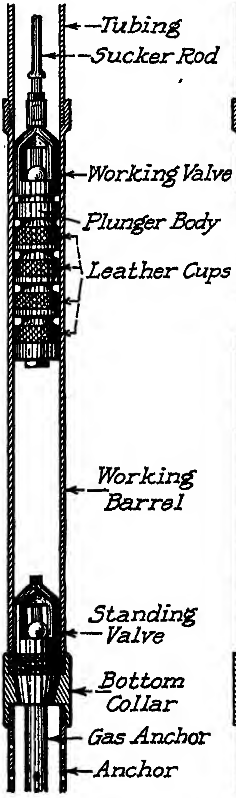

FIG. 1.—LEATHER-PACKED TYPE OF PLUNGER PUMP.

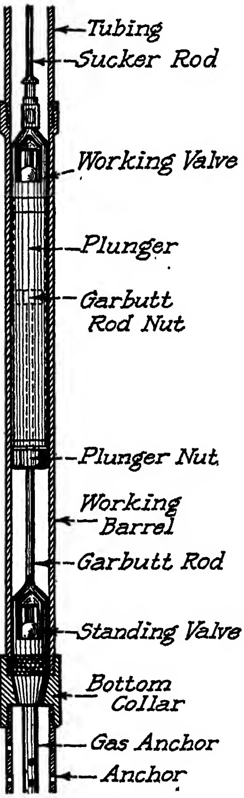

FIG. 2.—STEEL-PLUNGER TYPE OF PLUNGER PUMP.

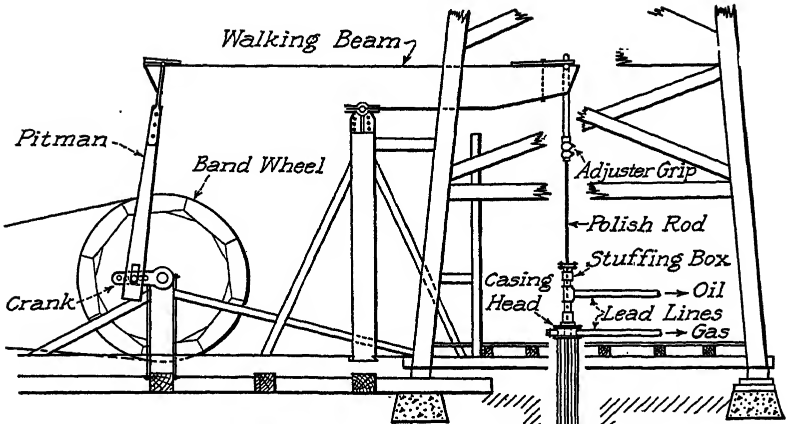
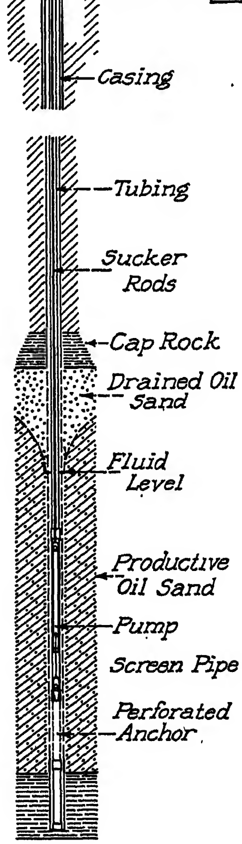

FIG. 3.—POWER CONNECTIONS FOR PLUNGER PUMPING.

rod" to facilitate removal of the standing valve with the plunger when repairs are necessary. The valves are usually of the ball and disk-seat type (see Fig. 4). The cup-leather packed plunger is suitable for use in shallow wells; but in deep-well pumping, where fluid pressures are high, or in cases where fine sand must be handled with the oil, the polished steel plunger is preferred.

The pump plunger is actuated by a column of "sucker rods" extending through the tubing on which the working barrel is suspended, from the top of the plunger to the surface. Just below the casinghead, the column of sucker rods connects with one end of a "polish rod," which operates

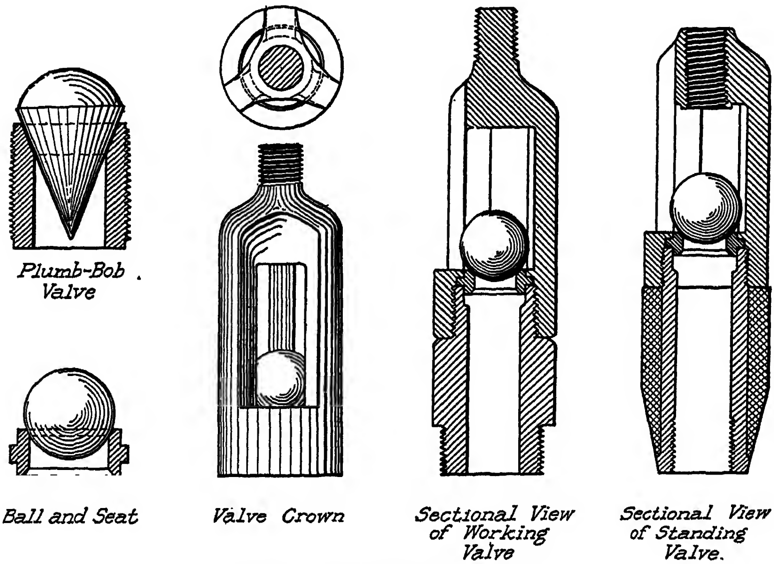

FIG. 4.—PLUNGER-PUMP VALVES.

through a stuffingbox in the top of the tubing, the upper end being suspended with the aid of an "adjuster grip" from the end of a walking-beam (see Fig. 3). A side outlet in the tubing, just below the stuffingbox, connects with the "lead line" through which the oil lifted by the pump is led to a gas trap, storage tank, or sump. An oscillating motion is imparted to the walking-beam by a pitman and crank actuated by a power-driven band wheel. A steam, gas, or oil engine or an electric motor may be the source of power. For the pumping of comparatively shallow wells operated in multiple by a system of pull lines from a central power station, a "pumping jack" is substituted for the band wheel-pitman-walking beam combination, but multiple pumping is not adaptable, generally, to wells exceeding 3000 ft. in depth, with which we are here primarily concerned.

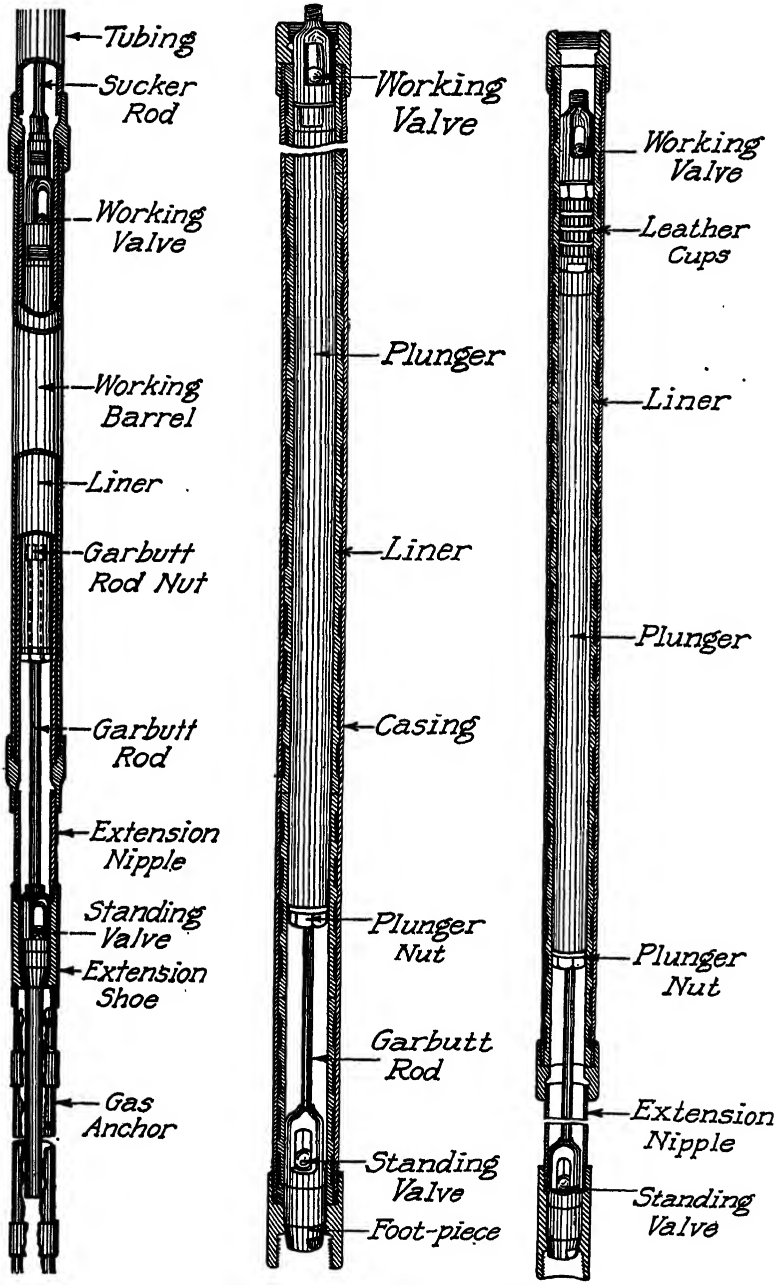

FIG. 5.—TYPES OF OIL-WELL PLUNGER PUMPS.

Working barrels are customarily made of cast iron, about ½-in. thick and ranging from 4 to 7 ft. in length, depending on the length of stroke desired. The bore of the working barrel is as nearly cylindrical as possible and is highly polished on its interior surface. The D. & B. type of working barrel is equipped with a liner of tubular steel, which can be readily replaced when the plunger clearance becomes too great as a result of wear of the polished surfaces. The Axelson working barrel has a liner made up of short separate and readily removable cast-iron sections, so that one part of the liner may be replaced without the necessity of renewing all parts.

Plungers are simple steel cylinders, usually about 3/16-in. thick, highly polished on the exterior surface and ground to a close fit with the inside polished surface of the working barrel. For shallow-well service, the plunger is often made up of leather cups mounted on a short steel body (see Fig. 1). Leather cups are also occasionally used in connection with long steel plungers in deep-well pumping (see Fig. 5); but if so used they should be placed between two steel plungers to prevent their

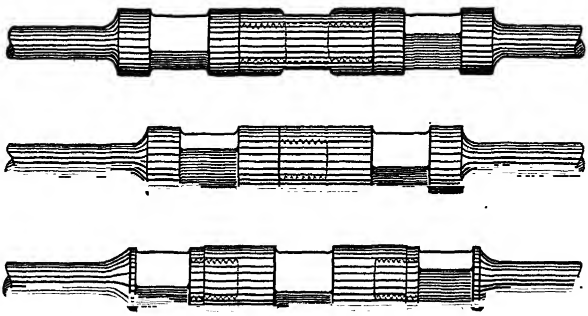

FIG. 6.—TYPES OF SUCKER-ROD JOINTS.

distortion under the excessive pressure. Sulfur in the oil will destroy leather cups in a short time, so that they must be frequently replaced.

Valves and seats are commonly made of hardened tool steel and the ball is carefully ground to fit the beveled interior edge of the disk-shaped seat. The latter is reversible so that either side may be placed uppermost The cage, or crown, enclosing the ball has either three or four wings and is threaded so that it may be rigidly attached to the top of the plunger, in the case of the working valve, or to the tapered body of the standing valve. For use in pumping oil carrying corrosive ground water, the pump valves and seats are occasionally made of brass or bronze; brass-lined working barrels are also used in such cases.

Sucker rods are of steel, cylindrical in form, and the 20 or 30-ft. lengths are connected by box-and-pin tapered joints of the form illustrated in Fig. 6. Diameters of the rods range from 9/16 to 1 in. The threaded ends are sometimes welded to the rods but a superior type of rod is made by

upsetting the ends of the rods, shaping the wrench square and joint reinforcement, and cutting threads directly thereon. In another type of sucker-rod joint, the rods are made alike at each end, the individual rods being connected by recessed and threaded couplings. This form of joint has the advantage that most of the wear in service falls on the couplings, which can be cheaply replaced without the necessity of cutting and welding the rods. The pump plunger may also be operated by a continuous wire cable connecting with the walking beam, but this form of power connection is less desirable than the solid rods in deep-well pumping because of its greater elasticity.

Tubing is of steel, somewhat heavier than standard pipe, and ranges in diameter from 1¼ to 4 in., depending on the size of pump. The couplings are longer than ordinary and are recessed at the ends for facility in connecting and disconnecting. For deep wells, upset-end tubing has greater strength.

Operating Cycle of Oil-well Plunger Pump

The plunger makes from 15 to 40 up-and-down strokes of from 24 to 60 in. per min. The speed is controlled by that of the engine or motor, while the length of stroke is adjustable by placing the wristpin connecting the crank with the pitman, in one of several holes in the crank, at varying distances from the center of rotation (see Fig. 3). The suspension mechanism for the polish rod is adjusted so that, in ordinary operation, the lower end of the pump plunger is never within 6 in. of the top of the standing valve.

Assuming that the pump plunger is at the lower extremity of its travel, as the up-stroke begins, reduction in pressure is effected in the space between the two valves, the lower valve opening and admitting oil through the standing valve into the working barrel. Influx of oil continues during the up-stroke of the plunger; but with the succeeding down-stroke, the standing valve closes and the oil in the barrel is subjected to pressure which lifts the working valve, the oil being displaced through the working valve as the plunger descends. The succeeding up-stroke of the plunger lifts the oil, thus displaced, into the tubing and draws a fresh supply of oil into the working barrel for a repetition of the cycle. Each successive stroke of the plunger forces additional oil into the tubing until it overflows into the lead line at the surface. It should be noted that the pump lifts oil only during half of its cycle, that is, on the up-stroke of the plunger.

Principles of Plunger Pumping

While the cycle of operation of the plunger pump is not complex and its working parts are simple and few, when applied in deep-well pumping,

various difficulties are experienced which lead in many cases to gross inefficiency and loss of production. A proper understanding of these losses necessitates a study of the principles of plunger pumping and of the difficulties experienced in operation and maintenance of the equipment.

Motion Afforded by Walking-beam

A study of the motion afforded by the crank, wristpin, pitman, walking-beam, and sucker-rod combination shows that it is well adapted to the purpose of operating the plunger type of pump, if the parts are properly proportioned and if the speed and length of stroke are adjusted to the depth of the well, submergence, and viscosity of the oil. Theoretically, if the crank revolves at constant angular velocity and if the pitman is infinitely long, the plunger of the pump moves in simple harmonic motion. The plunger is uniformly accelerated from a position of rest, at the upper end of its stroke, to a maximum speed, at the mid-point of the stroke; then uniform negative acceleration results in gradually bringing the plunger to rest at the lower end of the stroke. The up-stroke is accomplished with an equivalent sequence of movements. This is an ideal motion for plunger pumping, but it is seldom perfectly realized because of the tendency of the engine to increase its speed on the down-stroke of the rods, because the short length of the pitman contributes some eccentricity to the motion, and because the elasticity of the sucker rods does not permit the plunger to respond precisely to the movement of the walking-beam. Partial correction of these disturbing factors can be effected by counterbalancing the walking-beam so that the same amount of work is performed on the up-stroke of the rods as on the down-stroke, by using as long a pitman as possible and sucker rods of large diameter, so that the stretch of the rods is reduced to a minimum.

Sucker-rod Failures

Sucker-rod failures are due to many causes which result in overstressing, wear, and eventual parting of the column of rods connecting the pump plunger with the walking-beam. The rods are subjected to a combined static and dynamic load. The static load results from the downward pressure of the column of oil in the tubing above the pump plunger and the dead weight of the rods themselves. The dynamic load is equivalent to the force necessary to overcome the inertia of the long column of oil and rods that must be accelerated from a position of rest with each up-stroke of the pump, to a speed of from 80 to 800 ft. per min., depending on the length of stroke and number of strokes of the beam per minute. This resistance of the oil column to upward translation is augmented by the viscous drag of the oil on the metal walls of the tubing.

Sucker-rod Stresses

In deep-well pumping, only the larger sizes of rods are used because of the great stress put upon them. Assuming that 1-in. rods are used in a well 4000-ft. deep, operating a 2.75-in. plunger (3-in. pump), pumping oil having a specific gravity of 0.95, the static oil load will be 8480 lb. and the dead weight of the rods will add 11,400 lb., making a total weight for the oil and rods of 19,880 lb. To this must be added another load representing the force necessary to move the plunger in the working barrel. This will probably account for 100 or 200 lb. of additional static load if the clearance of the plunger is small enough for efficient operation under high pressure; say, 120 lb. to make the total static load a round 20,000 lb.

When estimating the dynamic load on the sucker rods, the force necessary to accelerate a mass equivalent to the total static load, from rest to the maximum upward velocity attained during the up-stroke, must be computed. If the pump makes twenty-five 36-in. strokes per minute, the acceleration will be 10.27 ft. per sec. per sec. and the force necessary to accomplish this is 6380 lb. To this must be added the force necessary to overcome the frictional resistance of the oil on the walls of the tubing, a force difficult to estimate precisely, but estimated at approximately 340 lb. for oil of 0.5 poise viscosity, flowing at a maximum velocity of 300 ft. per min. through 3-in. tubing. We thus have a dynamic load of 6720 lb., which, added to the static load, gives a total load on the upper sucker rods, at the maximum velocity of the up-stroke, of 26,720 lb. This, for a rod of 1-in. diameter, is equivalent to a unit stress of 34,021 lb. per sq. in. of cross section and represents a safety factor of only 1.8, in terms of the breaking strength of the metal; the steel is stressed almost to its elastic limit (about 35,000 lb. per sq. in.). It is to be noted that the more important of these static and dynamic loads vary directly with the depth of the well and that if the total load on the rods for a 6000-ft. well were computed for the conditions just assumed, the elastic limit of the steel would be considerably exceeded.

Strain on Rods Due to Sanding up of Pump

The greatest strain of all comes on the rods when the pump becomes clogged with accumulated sand and the plunger binds in the working barrel. At such times, the full power of the engine, with the additional mechanical advantage provided by the band wheel and crank, is brought to bear upon the rods; if the pump plunger is unable to respond, the rods snap, either under direct tensional strain on the up-stroke or by buckling and compressive strain on the down-stroke.

Fatigue of Rod Metal

Failure of the rods in pumping service is probably often due to fatigue of the metal. While the rods are supposed to be always in tension,

the tension on the down-stroke (equivalent to the weight of the rods and plunger less the friction of the plunger on the working barrel and the viscous drag of the oil in the tubing on the rods) is comparatively small in comparison with that prevailing on the up-stroke. This continual variation in tension, because of reversal of motion every 1.2 sec. (25 strokes per minute), is exceedingly destructive in its influence on the molecular structure of the steel. Investigations have shown that in reciprocating mechanisms of this class, the life of the steel in the connecting-rods is directly proportional to the number of reversals. Operating the beam more rapidly than the plunger can respond in the working barrel throws an added strain on the rods, in that the lower end of the column of rods may actually be under compression on the down-stroke. This results in more rapid fatigue of the metal and destructive bending stresses and causes buckling of the rods against the tubing, causing abnormal wear on both.

Wear of Rod Metal

Aside from the great tensional strain and fatigue to which the rods are subjected, they are also weakened in continued service by wear; the result of rubbing against the tubing and the scouring action of sand carried by the oil. Most of this wear falls on the enlarged sections at the joints, which may be remedied by upsetting and redressing, or by cutting off the worn end and welding on a new box or pin and wrench square. In the top joint illustrated in Fig. 6, most of the wear falls on the coupling, which can be readily replaced. Sucker rods often fail at defective welds, and particular attention is given by successful rod manufacturers to the condition of the metal in the vicinity of welds and at the forged and threaded end sections. Rods sometimes part in the well by unscrewing at some joint not properly made up, or the joints may unscrew as a result of excessive vibration if the motion is too rapid.

From the foregoing, it is evident that sucker-rod failures in deep-well pumping are caused primarily by overstressing of the steel as a result of the selection of rods too small in cross-section to resist the loads imposed, to the use of too rapid a stroke, to fatigue and loss of metal by wear in service, and to abnormal strains occasioned by "sanding up" of the pump. The remedy would seem to lie in the use of rods of larger cross section and slower pumping speeds and elimination of sand from the pump.

Tubing Difficulties

The tubing on which the working barrel is suspended is not subjected to such destructive influences as are the sucker rods, but tubing failures cause occasional interruptions in routine pumping operations. These are chiefly the result of wear, largely from the frictional contact of the

oscillating rods against the inner surface of the tubing. Holes are occasionally worn through the tubing by the rods so that leaks develop, or the tubing may be so weakened that it pulls apart under its own weight. Difficulties of this sort are particularly common in crooked holes, where the rods may rub continually on one side of the tubing, or where "whipping" of the rods results from too rapid a stroke, or where the plunger fits so tightly in the working barrel that it does not respond readily to the downward stroke of the rods.

Inefficiency of Pump

Inefficiency of the pump may be due to a variety of causes, the more common of which are the result of worn plungers, working barrels, and valves, the presence of sand or free gas in the pump, inadequate submergence of the standing valve, too rapid a movement of the plunger and elastic elongation of the column of sucker rods, which shortens the effective stroke. Because of these inefficiencies, the pump will often deliver less than half of the volume of oil indicated by the theoretical plunger displacement.

Valve Leakage

Valves and seats, though made of a special chilled steel, are often subjected to rapid and irregular wear, causing oil leakage under the high pressures prevailing in deep-well service. The efficiency of the pump is directly dependent on the security of the valves against oil leakage. Fine sand suspended in the oil is especially destructive and in some wells, in which the sand problem is particularly troublesome, valves and seats last but a few days. Aside from leakage of valves from actual wear of the metal balls and seats by the scouring action of sand, a few grains of sand caught between the ball and its seat will permit much of the oil to be forced back through the valve, with marked loss of efficiency.

Plunger Leakage

Wear of the polished surfaces of the working barrel and plunger will also permit oil leakage around the plunger and reduce the pump efficiency. New pumps intended for deep-well service are sometimes finished with a clearance between the working barrel and plunger of only 0.00025 in., but normal wear gradually increases this; if sand finds its way between the polished surfaces, destruction is rapid. The clearance between the plunger and barrel should be sufficient to permit the plunger to respond readily on the down-stroke of the rods, otherwise the rods immediately above will be placed under compression and buckling results, with attendant abnormal wear on both the rods and the tubing.

It is difficult to make a long working barrel or plunger that is absolutely straight and uniform in diameter, so that generally the wear is not

uniformly distributed. This effect can be minimized through the use of a rod rotor, a device attached to the walking-beam and polish rod, causing a part revolution of the rods and plunger with each stroke of the beam, thus distributing the wear caused by inequalities in alignment and diameter of the plunger and working barrel.[1] In the multiple-plunger pump, it is sought to correct this difficulty by using two, three, or four closely spaced, short plungers with flexible connections, instead of a single long plunger.[2]

Influence of Temperature on Plunger Clearance

A carefully made working barrel and plunger do not necessarily maintain their shop clearance when placed in service in the well. Prevailing ground temperatures at depths of 4000 or 6000 ft. are commonly much higher than normal atmospheric temperatures. Temperatures of from 125° to 175° F., which are not unusual in deep wells, will cause expansion of the plunger and working barrel, and the two, being usually of different materials and different thickness, will not expand to the same degree, so that the desirable clearance provided in the shop becomes greater or less than it should be for best results. Increase in the clearance space will result in greater leakage around the plunger, while decreased clearance throws an additional frictional load on the sucker rods. In extreme cases, the pressure developed between the two polished surfaces may prevent proper lubrication of the plunger so that it overheats, and the situation is aggravated to such a degree that the plunger binds in the barrel and the rods part.

Influence of Oil Pressure on Plunger Clearance

It seems probable that under deep-well conditions, the clearance of the plunger in the working barrel may also be somewhat altered by distortion due to oil pressure. The oil pressure developed between the polished surfaces of the plunger and barrel by the static head of, say, 4000 ft. of oil in the tubing, would be about 1600 lb. per sq. in., a force sufficient to accomplish perceptible compression of the plunger and expansion of the working barrel, thus increasing the clearance and inducing oil leakage. Placing the working valve at the bottom of the plunger instead of at the top will remedy this difficulty, as the oil pressure will then be applied within the plunger, expanding the thin metal cylinder until it is pressed tightly against the polished interior surface of the barrel. If a loose-fitting plunger is used, there will be but little resistance to movement

[1] L. Suverkrop: "Plunger Rotators," Summary of Operations, California Oil Fields (October, 1924); California State Mining Bureau.

[2] J. A. Zublin: "Pumping Deep Wells in California," Paper read before Fort Worth Meeting of American Petroleum Institute (December, 1924). *Oil Weekly* (Dec. 19, 1924) 35.

in the working barrel on the down-stroke, while the fluid pressure may be relied on to expand the plunger to a tight fit on the up-stroke.

Influence of Natural Gas on Pump Efficiency

Placing the working valve at the lower end of the plunger instead of at the top is also advantageous in that it reduces the space between the valves in which gas may accumulate. Oil drawn through the standing valve usually contains dissolved gas under pressure. Reduction of pressure, due to vacuum conditions within the working barrel on the up-stroke of the plunger, causes much of this gas to be forced out of solution, assuming the gaseous phase and becoming occluded or entrained in the oil. Some upward segregation of this gas from the oil in the working barrel occurs, even in so short a time as that consumed by the up-stroke of the plunger, so that when the plunger begins its down-stroke, the working valve is in contact with gas instead of oil. This gas must be compressed to the pressure of the superimposed column of oil in the tubing before the valve may open and permit upward displacement of the oil in the tubing; the effective length of stroke of the plunger is thus appreciably shortened. If a large amount of gas accumulates in the working barrel, the working valve may remain closed, the plunger merely expanding and compressing the gas between the two valves with each up- and down-stroke. Reducing the clearance volume between the two valves reduces the opportunity for a "gas lock" to form in this way and tends toward increased pump efficiency.

Large amounts of free gas may be excluded from the pump by the use of a gas anchor; this is a section of perforated pipe screwed to the lower end of the standing valve body (see Fig. 3). The perforations through which the oil is passed to the standing valve of the pump are placed at such depth below the surface of the oil in the well that very little gas may find its way into the pump. In the case of the Zublin gas anchor, the oil is admitted to the standing valve through a single round hole, care being taken to proportion the diameter of the hole to the size of the pump valves. Most of the gas, in separating from the oil, finds its way upward through the annular space between the tubing and the perforated liner and is permitted to flow to the surface between the pump tubing and the well casings.

Influence of Submergence on Pump Efficiency

Another important cause of pump inefficiency is failure to maintain adequate submergence. Oil flows through the standing valve to fill the working barrel by reason of the difference in pressure existing within and without the pump. The suction effect of the rising plunger can never exceed 14.7 lb. per sq. in.; if this is insufficient to draw the oil in rapidly enough to fill the working barrel, additional static pressure must be

provided within the oil by maintaining the fluid level of the well at an elevated position above the standing valve.

The submergence necessary for efficient results will vary with the viscosity of the oil, the frictional resistance offered by the valves and pump openings, and the speed and length of stroke of the pump. When it is remembered that the oil must lift the standing valve and flow through the restricted orifices of the pump in from 1 to 2 sec., the possibility of pump inefficiency resulting from failure of the oil to follow up into the working barrel as rapidly as the plunger ascends is readily understood. Many oil producers plan to maintain a submergence of about 50 ft. on the standing valve when conditions permit. Experiments have shown that a greater submergence than this is of little advantage, even in pumping heavy, viscous oils with small pumps operated at high speed. However, few operators have any precise information concerning the fluid levels in their pumping wells and gross inefficiencies doubtless result in many cases from lack of knowledge of this important factor.

The static pressure of a column of oil in the well sufficient to provide reasonable submergence for the pump will often oppose the influx of oil from the reservoir sand to such a degree that the productivity of the well is greatly reduced. Determination of the most advantageous fluid level thus requires consideration of two opposing factors: low fluid level results in greater influx of oil into the well but low pump efficiency, and vice versa. The author has shown[3] that the fluid level which permits a maximum yield of oil from the sand may be approximately computed, and has proposed the use of a fluid-level indicator designed to show the position of the oil surface in the well while the pump is at work. Unless the producing sand is thick, it may be impossible to maintain the most desirable fluid level from the standpoint of securing maximum flow from the reservoir sand, without sacrificing pump efficiency through insufficient submergence. In such cases, if the formation immediately beneath the oil sand does not contain water, it is often desirable to drill a pocket beneath the oil sand into which the pump may be lowered when low fluid levels are desirable.

Influence of Plunger Velocity on Pump Efficiency

Within the limits determined by the physical ability of the oil to flow through the standing valve and follow the plunger in its upward movement, the capacity of the plunger pump is determined by the rate of plunger travel. An upward plunger displacement of 120 ft. per min. will pump twice as much oil as would be lifted with an upward plunger travel of 60 ft. per min. This relationship, however, is influenced by the compressibility of the oil itself, which operates to retard prompt response

[3] The Significance of Fluid Level in Oil-well Pumping, issued as Paper No. 1410-G, with Mining and Metallurgy, February, 1925.

of the valves, thus shortening the effective stroke. The plunger speed must be adjusted so that the valves have time to function properly and so that the plunger may sink through the oil in the working barrel on the down-stroke, without buckling the sucker rods. It is not only essential that the plunger speed be kept within reasonable limits, but the number of reversals must be reduced to a minimum; that is, the inefficiencies resulting from reversal in tension of the rods and operation of the valves are multiplied directly by the number of strokes. Such considerations demonstrate conclusively the advantage of a long, slow stroke.

As the pump efficiency falls off as the rapidity of stroke increases, there must be a certain rate of plunger velocity that will give maximum production of oil. This must be determined by the operator for each individual well, and if the pump is incapable of keeping pace with the potential productivity of the well, a pump of larger diameter should be installed, or a longer, slower stroke provided. Pump diameters are usually limited by the free working space within the well casing, and where it has been necessary to secure additional pump capacity, operators have been working toward a longer stroke. Several long-stroke pumping devices, attachable to the crank-shaft and pitman or to the walking-beam, to give the rods a longer stroke, have found practical application in deep-well pumping within the last year or two. They provide a pumping stroke of 72 in. in some cases, instead of the usual 24 to 36 in. Working barrels must necessarily be longer when long-stroke pumping is practiced, an 11-ft. barrel being desirable for a 72-in. plunger stroke. One long-stroke pumping device actuated by hydraulic pressure, recently applied in a few deep wells in the California fields, accomplishes ten 12-ft. strokes per minute. When long-stroke pumping is practiced, the angular displacement of the polish rod makes the use of a walking-beam yoke desirable. Such a yoke serves to keep the polish rod in alignment and provides a straight upward pull on the rods for all positions of the beam.

Influence of Valve Design on Pump Efficiency

The design of the pump valves to accomplish more positive action, to distribute wear, and to reduce resistance to movement of the oil has received considerable attention on the part of pump manufacturers. Fig. 4 illustrates the usual type of valve—a polished steel ball on a bevel-edged, reversible, annular, disk-shaped seat. This valve offers considerable resistance to movement of oil through it, for if the valve is lifted but a short distance off its seat, the oil must abruptly change its direction of flow and force its way through a restricted, annular-shaped opening surrounded by angular corners and recesses, which induce eddy-currents and encourage the release of gas from the oil and the formation of emulsions if water is present. Some manufacturers are featuring so-

called "over-size" valves of the same type, which provide a larger valve opening and therefore reduce these difficulties somewhat. There has also been a tendency toward the introduction of radically different types of valves, such as the plumb-bob valve (also shown in Fig. 4), and the Parker valve, a beveled, disk-shaped valve mounted on a cylindrical stem. However, these special forms have found but limited use and do not show promise of replacing the spherical type.

When the working valve is placed on the lower end of the pump plunger, it is necessary to eliminate the garbutt rod. This also reduces the resistance to flow of the oil, particularly where it passes through the lower end of the plunger, and increases the plunger displacement, hence the pump capacity. In pumps having this feature, other means must be provided for removing the standing valve with the plunger when repairs are necessary. For this purpose, a special locking device can be provided on the lower end of the plunger, which makes it possible to engage the crown of the standing valve. The "standing valve puller," as this device is called, engages the crown of the standing valve when the plunger is lowered and turned to the right by turning the rods from the surface.

Influence of Elastic Elongation of Sucker Rods on Pump Efficiency

Elastic elongation of the sucker rods during the up-stroke of the plunger will result in serious reduction in the effective length of stroke in deep wells. In the case of 1-in. rods, the tensile stress on the up-stroke was found to be about 34,000 lb. per sq. in. of cross section. This is the maximum stress developed in the rods near the surface that must support the dead weight of the entire column. At the lower end, near the pump plunger, the stress would be much less—only 19,270 lb. The average stress for the entire column on the up-stroke would, therefore, be about 26,600 lb. Similarly, the average tensile stress for the down-stroke of the pump is computed as approximately 7100 lb. per sq. in.; hence, the net increase in stress with each reversal of stroke will be about 19,500 lb. per sq. in. Computation of the elastic elongation of a 1-in. steel rod, 4000 ft. long, under this stress, indicates an elongation of about 31 in., which means that if the polish rod is given a stroke of 36 in., the effective stroke of the plunger will only be 5 in., representing a direct loss of efficiency of 86 per cent. A 48 or 60-in. stroke will give greater efficiency.

Influence of Sand Influx on Pump Efficiency

Sand entering the well with the oil is detrimental from every point of view. If it finds its way into the pump, it rapidly abrades the working surfaces and may accumulate in the oil passages so that movement of the oil through the valves is entirely prevented, necessitating withdrawal and cleansing of the rods, tubing, and pump. If the sand accumulates in the well about the pump, it materially restricts the free access of oil from

the reservoir rock to the well and to the pump. Sand should be excluded from the well by every possible means.

If influx of sand is a serious problem, it may be necessary to maintain abnormally high fluid levels in order to create additional back pressure on the producing stratum and restrain the sand movement. In pumping fluid from unconsolidated sands, working barrels are often suspended hundreds of feet above the source of the oil, the well being compelled to produce against the static pressure of the intervening column of oil. To prevent the fluid level from falling to a point where sand influx would be imminent, the oil holes in the gas anchor are so placed that the pump will suck air when the oil surface approaches the critical level. In such cases, the pump merely skims off the surface oil and is fairly free of sand, but the flow of oil from the sand is greatly reduced in comparison with what would be possible if it were practical to maintain a lower fluid level.

The development and use of sand-control devices either to effectively screen the sand from the well without reducing the drainage effect, or to permit the pump to take care of large quantities of sand in suspension in the oil without clogging or excessive wear of the pump parts, would make possible increased production of oil through the maintenance of more appropriate fluid levels. The various types of screens available on the market are only partly effective in excluding sand from the pump. The author has proposed the use of a gravel-filled cavity about the well within the oil sand as a means of increasing production and reducing sand incursion.[4] If the size of the gravel particles is properly proportioned with respect to the size of the sand grains, the latter bridge over the apertures between the gravel particles and the main mass of gravel and the well pump at its center remains free from sand incursion. The writer has also patented a process and apparatus for forming such a cavity about the well and filling it with gravel.[5] Several devices are employed in connection with oil-well pumps, which are intended to enable them to separate the sand from the oil or to pump the sand to the surface along with the oil. One of these, called an "anchor dump," is designed to drop the sand out of the working barrel as it settles in the gas anchor below the standing valve. The manufacturers of several oil-well pumps claim that their particular pumps are especially adapted to the handling of oil carrying sand in suspension, by reason of certain peculiarities in design.

Much may be accomplished in avoiding the effects of accumulated sand, by designing the pump to avoid changes in oil velocity. Enlarged recesses above the standing valve are particularly apt to accumulate sand, which eventually clogs the valve. The carrying capacity of oil for sand varies as the square of its velocity; hence, by building pumps with an oil

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[4] Increasing the Production of Petroleum by Increasing the Diameter of Wells. Issued as Paper No. 1364-G with Mining and Metallurgy, October, 1924.

[5] Patent No. 1530221, issued March, 1925.

channel of slightly smaller cross section than usual, the carrying capacity of the oil for sand will be greatly increased. There is some advantage, when sand must be pumped with the oil, in using a long working barrel together with a plunger no longer than the length of stroke employed. This combination permits the plunger to operate without its upper end extending out of the working barrel, so that sand grains settling about the working valve have little opportunity to find their way between the plunger and the barrel; and as the plunger is no longer than the stroke, any sand that may be dragged in between the two surfaces at the top will have an opportunity to escape at the bottom during the same stroke and will not score the polished surfaces by continued abrasion.

Influence of Wax Accumulation on Pump Efficiency

Difficulty is also experienced in pumping paraffin oils, in the deposition of wax from the oil in the tubing, pump openings, screens, and perforations. This is probably due, in large part, to the cooling effect of expanding gas, particularly in and near the pump where most of the gas expansion occurs. The chilling effect of expanding gas must also result in a considerable increase in viscosity of the oil, leading to lower pump efficiency.

Influence of Gas Pressure on Oil Drainage from Sands

Still another important consideration in determining the overall efficiency of oil production from wells by plunger pumping is the mainte-

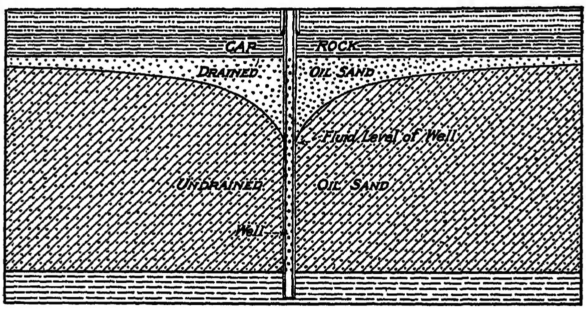

Fig. 7.—Illustrating formation of oil-drained spaces beneath the cap rock.

nance of gas pressure in or on the oil in the sand. It has been shown that gravitational segregation of oil and free gas in the sand reservoir leaves oil-drained spaces beneath the cap rock in which gas may accumulate if prevented from escaping through the well[6] (see Fig. 7). The downward pressure of this gas on the oil in the reservoir sand creates an equal hori-

[6] L. C. Uren: Elements of the Oil Well Spacing Problem, *Bull.* Amer. Ass. of Pet. Geol. (1925) 9, 193.

zontal pressure within the oil mass, causing the oil to flow more rapidly through the sand toward the well than it would if no gas pressure existed. Laboratory studies have shown that a gas pressure of only 10 lb. per sq. in. maintained above the oil causes the latter to flow from the sand many times as fast as is possible by simple gravity drainage without the aid of gas pressure (see Fig. 8). While this free gas greatly stimulates the flow of oil from the sand into the well, if it follows the oil and is taken into the pump, it becomes detrimental in that it reduces the efficiency of the pumping mechanism, as shown in a previous section. Every effort should therefore be made to keep the gas in the oil sand and out of the pump.

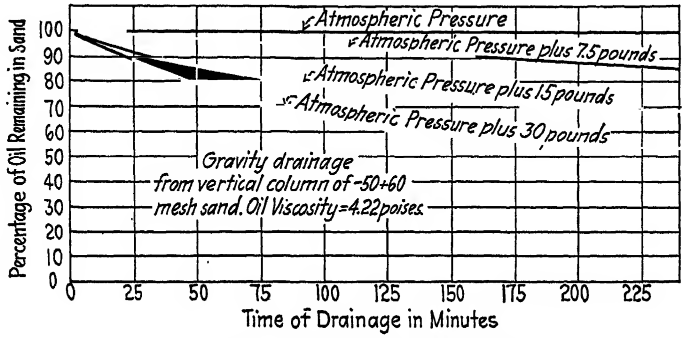

FIG. 8.—RESULTS OF PRESSURE-DRAINAGE TESTS FOR DIFFERENT HYDROSTATIC HEADS (50 TO 60 MESH SAND, 19° BÉ. RESIDUUM FUEL OIL).[7]

METHODS OF SECURING INCREASED EFFICIENCY IN PLUNGER PUMPING

Summarizing the principles discussed in the foregoing sections, for efficient extraction of petroleum from wells by pumping we should:

1. Use a long and slow plunger stroke and a pump of sufficient capacity to keep pace with the maximum rate of influx of oil from the sand, without excessive plunger speed.
2. Adjust the loads on the walking-beam so that the work performed on the up- and down-strokes of the beam will be equalized.
3. Use a long pitman so that its angularity with the crank circle will not cause excessive variation in acceleration of the pump plunger.
4. Use sucker rods of such diameter that elastic elongation under the variable strains imposed will not be excessive.
5. Use valves of large cross section and avoid pumps designed with circuitous oil channels interposing abnormal resistance to flow.
6. Place the traveling valve at the lower end of the plunger instead of at the top.
7. Provide plunger clearance that will be correct at the temperature and pressure to which the pump is subjected in the well.

[7] From thesis of R. S. McIntyre, University of California, 1924.

8. Place the pump as deep in the well as may be necessary to secure adequate submergence for satisfactory pump efficiency without maintaining fluid levels detrimental to the securing of maximum flow of oil from the reservoir rock.
9. Exclude float sand from the well and from the pump.
10. Keep the gas in the oil sand and out of the pump.

Improved Type of Plunger Pump for Oil-well Service

With the purpose of developing a pump and a system of pumping that would fulfill the requirements enumerated above, the writer has designed a pump that embodies a number of novel features. It operates efficiently under low submergence and has high capacity, so that low fluid levels can be successfully maintained. It provides a length of stroke double that of the ordinary oil-well plunger pump without increasing the walking-beam or sucker-rod movement. For wells of normal productivity, it may therefore be operated at half the usual speed without diminution of output. The pump is double-acting, lifting oil on both the up- and down-strokes of the beam and the mechanism may be perfectly balanced so that the engine performs the same amount of work on both strokes. It permits of ready separation of such sand as tends to enter the pump, so that sand does not find access to the plunger, working barrel, and traveling valves. Finally, it accomplishes separation of the gas from the oil in the well before the fluid enters the pump, forcing the gas back into the oil sand to perform further service in moving more oil into the well. By conservation of gas in this way, the ultimate production of the well and the percentage extraction of petroleum from the sand are increased, and the pump inefficiency resulting from the presence of gas in the working barrel is minimized.

For successful operation of the pumping system herein proposed, it is essential that the fluid level be maintained in the well below the cap rock, so that a body of partly drained oil sand may exist about the well in the upper part of the productive stratum, as indicated in Fig. 7. The fluid level so maintained should be that which will result in maximum flow of oil into the well from the reservoir sand, irrespective of considerations of pump efficiency as influenced by the submergence afforded. The pumping device creates its own necessary fluid level for efficient operation in an interior compartment, that is quite independent of the level of the fluid in contact with the walls of the well.

In the operation of the usual type of oil-well plunger pump, the working barrel is stationary in the well while the pump plunger reciprocates. The system of pumping herein proposed differs from that normally employed in oil-well pumping, in that both the tubing on which the working barrel is suspended and through which the oil rises to the surface and the rods that actuate the plunger are reciprocating, the two being

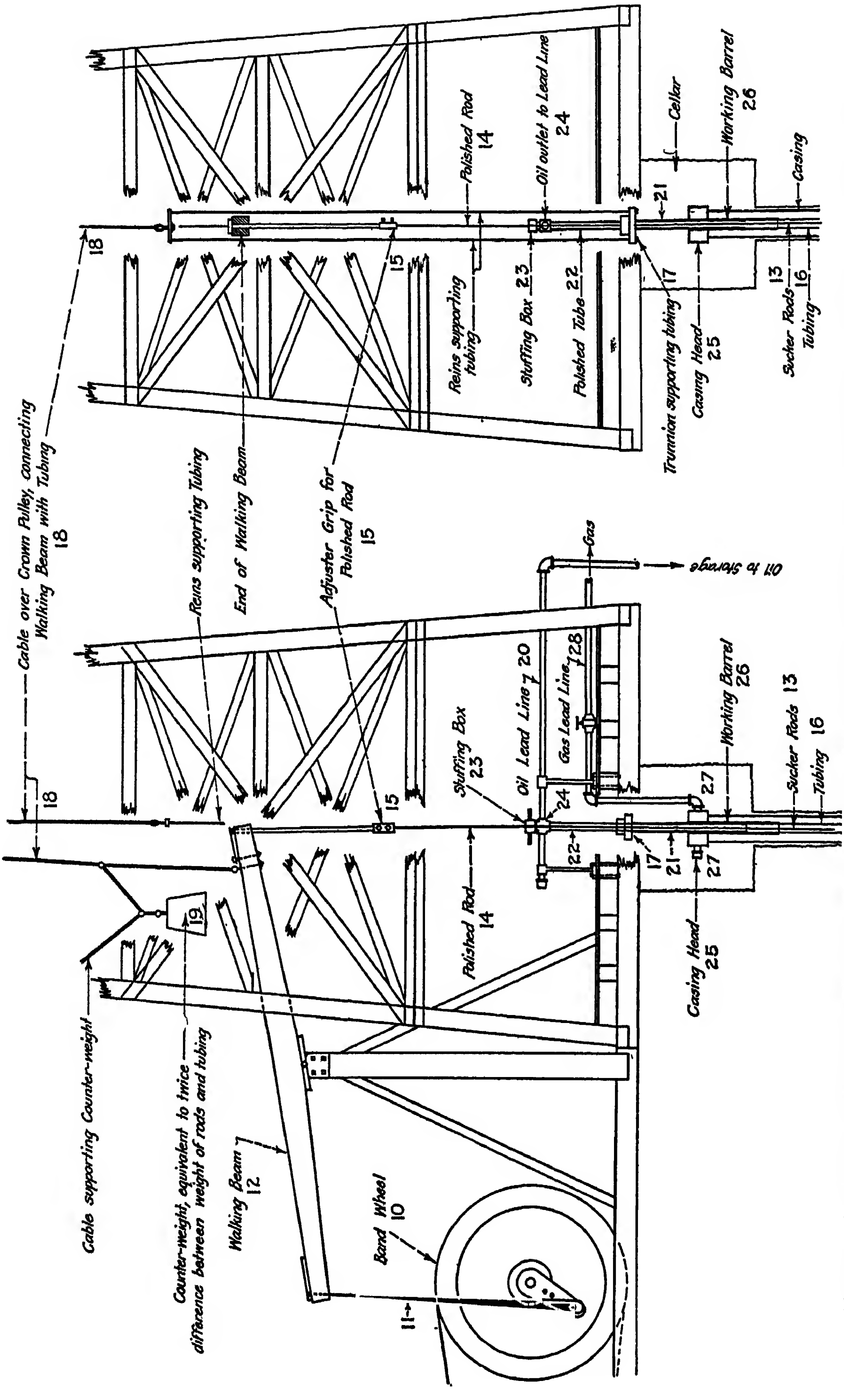

FIG. 9.—POWER CONNECTIONS AND SURFACE ARRANGEMENTS FOR OPERATING DOUBLE-ACTING OIL-WELL PLUNGER PUMP.

connected with the walking-beam in such a way that they move in opposite directions; that is, the rods are descending while the tubing is ascending, and vice versa, each moving vertically in the well through whatever length of stroke the beam provides. Fig. 9 shows the means by which the necessary motion of the rods and tubing may be secured. The usual equipment provided at an oil well for pumping service is illustrated, with a few additional features to accomplish the reciprocating movement of the tubing. The band wheel 10 is revolved by a belt drive from the engine or motor pulley; by means of the pitman 11 and the walking-beam 12 the revolving motion of the wheel is converted into a vertical reciprocating movement, which the end of the walking-beam overhanging the well transmits to both the tubing and the sucker rods. The column of sucker rods 13 is suspended from a polish rod 14, which, in turn, is suspended from the end of the walking-beam with the aid of an adjuster grip 15. The tubing 16 through which oil rises to the surface, and on the lower end of which the working barrel is attached, is suspended on a swinging trunnion 17, the long reins of which (one on either side of the walking-beam) permit the latter to oscillate without interference. The swinging trunnion is supported by a steel-wire cable 18, which passes over a sheave at the summit of the derrick, thence down to the end of the walking-beam overhanging the well, to which it is attached by a suitable bolt and hook. By this device the column of sucker rods in the well is, in effect, balanced against the column of tubing. The tubing, however, will normally be somewhat heavier than the rods, so to effect an approximate balance, a counterweight 19, suspended from one corner of the derrick, is attached to the beam end of cable 18. If a proper balance of the tubing and rods is effected, the walking-beam will only be required to provide a force sufficient to lift the oil and to overcome frictional resistance and inertia of the oil and of the moving parts of the pump. It is apparent that this arrangement also accomplishes the necessary relative movement of the plunger and working barrel, that is, while the tubing and working barrel are descending, the sucker rods and plunger will be ascending, and vice versa.

In order that stationary lead-line connections 20 may be made between the well tubing and the oil storage, the reciprocating tubing 16 is equipped at its upper end with a working barrel 21 having a polished, oil-tight interior surface (see Fig. 10). This working barrel fits snugly over a stationary oil-delivery tube 22, rigidly supported above the derrick floor. In the upper end of this stationary oil delivery tube, 22, the usual polished-rod stuffingbox 23 and tee, or cross connection, for the lead line 24 are provided. The space between the well tubing and conductor casing is closed by a gas-tight casinghead 25, also equipped with a polished working barrel 26, through which the reciprocating tubing operates without leakage of gas. Side outlets 27 of the casinghead permit

the flow of gas when valve 28 (Fig. 9) is opened, but normally the gas outlets will be kept closed in order to conserve gas.

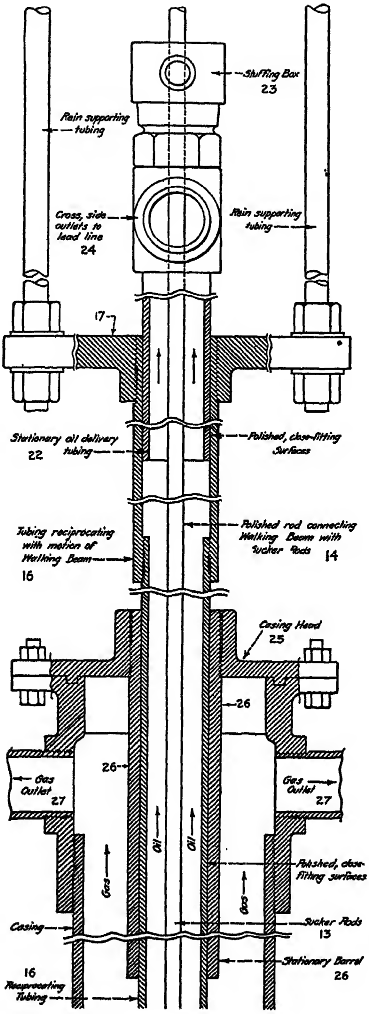

FIG. 10.—DETAIL OF ARRANGEMENTS AT CASING HEAD FOR OPERATING DOUBLE-ACTING OIL-WELL PLUNGER PUMP.

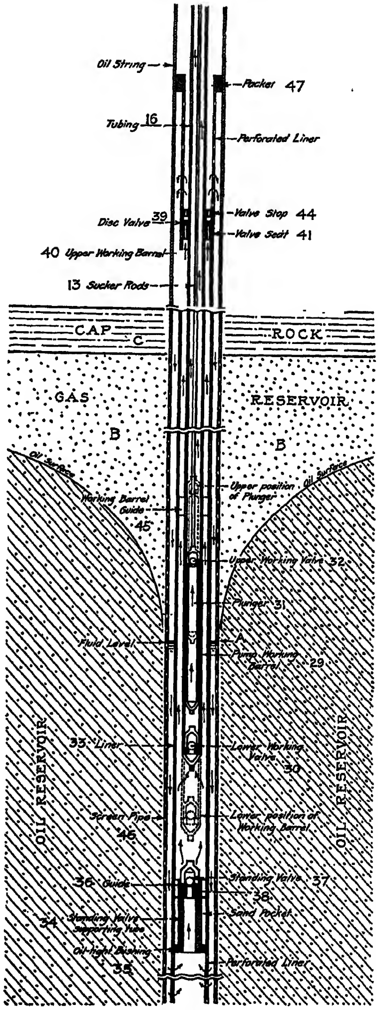

FIG. 11.—ASSEMBLY VIEW OF DOUBLE-ACTING OIL-WELL PLUNGER PUMP, ILLUSTRATING MANNER OF SEGREGATING GAS FROM OIL IN WELL.

The tubing and sucker-rod columns are continuous from the power connections just described to the pumping device, which is placed below the fluid level of the well. An assembly view of the various parts of the pumping device is given in Fig. 11 and details of the several working parts

are to be found in Figs. 12 and 13. A long working barrel 29 (Fig. 11) is suspended on the lower end of the tubing 16, forming virtually a continuation thereof. The working barrel is provided with a working valve 30, within its lower end, and has a polished interior surface that fits snugly over the polished exterior surface of the plunger 31 supported on the lower end of the column of sucker rods, 13. In either the lower or the upper end of the plunger, the upper working valve 32 is placed. The plunger and working barrel slide freely upon each other as they are actuated by the walking-beam and the intervening sucker rods and tubing.

The pump working barrel 29 is contained within an outer tube, or liner, 33, of such internal diameter that there is free space about the working barrel into which oil may flow and accumulate. A smaller concentric tube 34 is supported in a vertical position in the lower end of the liner by a bushing 35, providing an oil-tight connection between the two. The space between the two tubes serves as a sand trap into which any sand entering the liner may settle without clogging the valves and other moving parts of the pump. A guide 36 near the upper end of this annular opening keeps the smaller tube properly centered within the outer liner. The upper end of the tube 34 is fitted to receive a standing valve 37, which is merely pressed downward into position and held by the frictional contact of leather or fiber packing 38. Valves 30, 32, and 37 pass oil upward but resist downward movement.

At some distance above the pump working barrel, depending upon the gas pressure within the oil-producing formation, a valve 39 is attached to the exterior of the well tubing 16. This valve may be of the disk, or other type, operating within the annular space between the tubing 16 and the liner 33. The liner at this point is made somewhat thicker than elsewhere and is polished on its interior surface to form a working barrel 40 for the valve 39. This barrel must be somewhat longer than the maximum pumping stroke of the tubing. Though any type of valve that opens upward and resists the downward passage of oil and gas may be used for valve 39, Fig. 12 illustrates one of disk type in which the annular disk 39 is pressed down on the annular seat 41 on the up-stroke of the tubing; but on the down-stroke of the tubing oil or gas compressed below the valve causes it to rise from its seat, displacing the surplus gas and oil through the port openings 42. Such a valve must be prevented from turning, as by lugs 43, and cannot rise farther above its seat than the perforated annular valve-stop 44 permits.

The lower end of the pump working barrel 29 is held concentrically within the liner 33 by a centering guide 45 (Fig. 11) through which the barrel reciprocates freely. The lower end of the liner below the bushing 35 and the upper end above working barrel 40 are perforated to permit of the passage of oil and gas into the annular space between the liner and the screen pipe 46 with which the well is cased within the oil-producing

stratum. The space between the liner 33 and the screen pipe 46 is closed by a suitable packer 47 to the top of the liner, which, for best results, must be above the fluid level of the well.

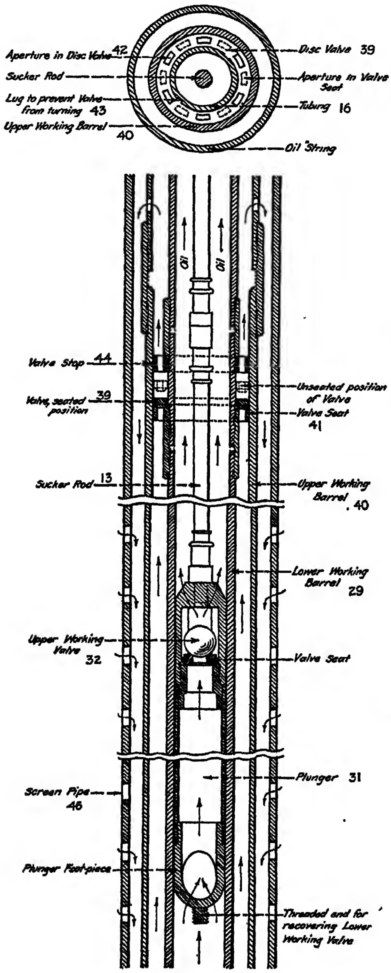

FIG. 12.—DETAIL OF UPPER END OF DOUBLE-ACTING OIL-WELL PLUNGER PUMP.

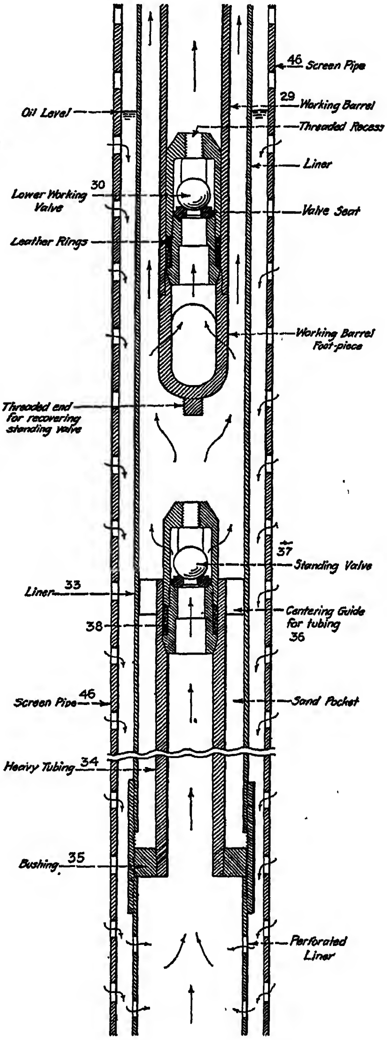

FIG. 13.—DETAIL OF LOWER END OF DOUBLE-ACTING OIL-WELL PLUNGER PUMP.

Operation of Double-acting Plunger Pump

Assume that the operating fluid level of the well for the particular stroke and speed at which the pump is operated is at *A*, Fig. 11, and that the oil level has been depressed some distance below the top of the pro-

ductive stratum by continued pumping, so that there is a pocket of drained oil sand *B* just below the cap rock *C* in which gas may accumulate. Assume that all gas outlets from the well are kept closed and that this pocket is charged with gas evolved from the oil in the underlying portion of the producing stratum under a pressure of, say, 50 lb. per sq. in. Because of this gas pressure, the fluid level *A* is depressed about 125 ft. below what it would be if the gas were permitted to escape freely under atmospheric pressure. Under the influence of this pressure, oil tends to flow rapidly through the perforations in the lower part of the liner 33 and up through standing valve 37 seeking its level within the liner.

Next assume that the walking-beam is set in motion, giving both the tubing and rods a vertical reciprocating movement of, say, 3 ft. As a result of the power connections provided, the pump working barrel 29 descends as the plunger 31 ascends, and vice versa. At one end of the stroke, valves 30 and 32 are, say, 6 ft. apart, while at the other end of the stroke they will be 12 ft. apart; *i. e.*, a displacement equivalent to double the pumping stroke of the beam. As the tubing rises on its up-stroke, valve 39 closes, creating suction within the liner 33, causing valve 37 to open and drawing oil up into the space about the working barrel 29. The fluid level within the liner 33 will rise as successive strokes of the valve 39 evacuate the gas or air enclosed within the liner above the oil surface, until a head of oil within the liner above the outside fluid level, equivalent to the full gas pressure within the well plus one atmosphere, is attained. Thus, if the gas pressure is 50 lb. per sq. in., as assumed, the oil will rise within the liner to a height equivalent to 64.7 lb. per sq. in. For oil having a specific gravity of 0.95, this will be about 160 ft. In order that valve 39 may have the lifting force of oil rather than gas operative upon it in normal operation, however, it is preferable that it be placed with its working barrel 40 at a slightly lower elevation. After the liner is filled with oil, valve 39 will discharge with each downward stroke, a small quantity of oil into the space within the barrel above the valve, whence it is permitted to flow back into the well through perforations provided at this point in the liner. This overflow of oil will carry with it such occluded gas as may escape solution in the oil during its entry into and passage through the liner. The reciprocating tubing operating valve 39 in the upper end of the liner has the effect of maintaining reduced pressure within the liner, so that a larger part of the gas that is held in solution in the oil at 50 lb. pressure is released and assumes gaseous form, rising through the liner and escaping with the oil overflow through valve 39. Any free gas that may be drawn with the oil through standing valve 37 follows a similar path. As a result of this action of valve 39, oil is taken into the lower working valve of the pump practically free of gas, and the gas liberated from the oil is returned to the free space in the well and to the oil sand above the established oil level.

Considering next the operation of the pumping device that lifts the oil to the surface, with reference particularly to Fig. 11, as the working barrel 29 descends, the plunger 31 will rise, both through a distance equal to the stroke of the walking-beam. During this movement, which is accomplished by the up-stroke of the beam, valve 32 in the plunger remains closed; while under the influence of the suction developed by the ascending plunger, valve 30 in the lower end of the working-barrel opens, admitting oil to fill the entire space within the working barrel and plunger between valves 30 and 32. On the succeeding down-stroke of the walking-beam, the plunger 31 will descend and the working barrel 29 will ascend. With this stroke, as a result of compression of oil within the working barrel and plunger, valve 30 is tightly pressed against its seat while valve 32 opens and passes the oil upward into the tubing on which the working barrel 29 is suspended from the surface. Each succeeding stroke of the pump lifts oil in the tubing until it overflows into the oil lead line 20 at the surface. The action of the pumping device is similar to that of an ordinary oil-well plunger pump, except that through the reciprocatting motion of the working barrel, in addition to that of the plunger, the pump has in effect double the stroke of a similar pump in which only the plunger reciprocates. The pump is also double-acting in that the working valve 30 lifts oil on the down-stroke of the walking-beam, while the plunger valve 32 lifts oil in the tubing on the up-stroke of the walking-beam. Such a pump would therefore lift approximately twice the amount of fluid delivered by a single-acting pump of equivalent beam stroke. It is apparent, also, that if the tubing and rods are balanced against each other, as herein proposed, the engine or motor operating the pump will have an approximately uniform power torque on both the up- and down-strokes of the walking-beam: a load equivalent to the column of oil superimposed on the pump valves plus that necessary to overcome friction and inertia of the oil column and moving parts of the pump and power connections.

In maintaining an oil-well pump in efficient operating condition, it is occasionally necessary to replace valves, valve seats, and other wearing parts. The apparatus herein described is designed to facilitate such removal and replacement of parts. The lower end of the pump plunger 31 may be equipped with a threaded connection or standing-valve puller to engage the cage of valve 30, which is held in its position in the lower end of the pump working barrel by suitable packing, fitting snugly into a conical recess. Valves 30 and 32 with the pump plunger 31 may thus be withdrawn to the surface on the end of the column of sucker rods. Valve 30 may be pressed back into its recess and the plunger raised to working position by reversing the process. Standing valve 38 is also mounted in a conical recess with packing rings providing a frictional hold in the top of tube 34, and may be engaged and withdrawn

to the surface with the pump working barrel 29, by raising and uncoupling the well tubing, which also brings valve 39 to the surface. The liner 33 and its working barrel 40 will receive but little wear, but these, too, may be engaged and withdrawn to the surface by a suitable fishing tool after releasing the packer 47.

In some wells, sand enters the pumping device with the oil and causes trouble by clogging and scouring the valves and valve seats. To partly alleviate this difficulty, the annular pocket below the standing valve 37, between tubes 33 and 34, is provided. Any sand passing through valve 37 comes temporarily to rest within the liner 33 and sinks into and accumulates within the said sand pocket. This sand pocket may be occasionally cleaned of sand, without withdrawing the rods or tubing, by simply engaging the cage of the standing valve 37 with the lower end of the pump working barrel 29, lifting valve 37 out of its recess in the upper end of tube 34 and churning the working barrel up and down in the liner. The agitation of the sand and fluid, occasioned by the alternate suction and pressure within the lower end of the liner, will, except perhaps in aggravated cases, lift the accumulated sand into suspension in the oil, whence it may settle through tube 34 into the sump at the bottom of the well. Valve 37 may then be pressed back into working position, released from the pump working barrel, and pumping resumed.

Summary

In this paper an effort has been made to acquaint the reader with the difficulties encountered in deep-well pumping and to suggest the means by which the efficiency of the oil-well plunger pump may be improved. For those not familiar with the details of construction and operation of this type of pump, the essential parts and their functions are briefly reviewed. The principles of plunger pumping are next discussed, the difficulties and causes of inefficiency encountered in practice are described and a code of rules for securing maximum efficiency enumerated. As a means of realizing the benefits that accrue to the oil producer through the observance of these principles, a type of pump is described that embodies several novel features, the use of which it is believed will result in greater mechanical efficiency, lower cost of production, and conservation of natural gas with attendant increased recovery of petroleum.

Influence of Submergence on the Efficiency of the Oil-Well Plunger Pump*

By L. C. Uren,† and V. J. Collins,‡ Berkeley, Cal., and S. B. Sargent, Tulsa, Okla.

[The conclusions arrived at by authors in paper read before the Petroleum Division and the California Sections of the American Institute of Mining and Metallurgical Engineers and the Standardization Division of the American Petroleum Institute, at a joint session held at Los Angeles, Cal., Jan. 21, 1926, are presented below.]

From the theoretical discussion and experimental evidence presented in this paper, the following conclusions may be drawn:

1. The efficiency of the oil-well plunger pump is dependent to an important degree upon the submergence provided.

2. Ample submergence must be secured in order to provide sufficient static pressure to force the oil through the restrictions interposed in the oil path at the standing valve and at the plunger nut surrounding the garbutt rod. These restrictions should be no smaller than is structurally necessary, in order to reduce resistance to oil flow to a minimum.

3. A part of the static pressure developed below the standing valve by submerging the pump is dissipated in overcoming the inertia and in providing upward acceleration for oil left between the pump valves at the conclusion of the down-stroke of the plunger, and of oil below the standing valve in the gas anchor above the oil inlet. This loss in effective static pressure can be reduced to a minimum by dispensing with the garbutt rod and placing the working valve at the lower end of the plunger. The valves should be as near together at the conclusion of the down-stroke of the plunger as is consistent with security in avoiding striking of the lower end of the plunger on the standing-valve crown. Gas anchors should be short, and oil inlets should be as near the standing valve as is consistent with proper gas exclusion.

4. Because of the presence of gas in solution in the oil, which tends to escape from solution under the reduced pressure conditions above the standing valve, the effective length of stroke of the plunger is reduced, and the suction effect of the rising plunger is largely nullified. The influx of oil through the standing valve is therefore chiefly dependent on static pressure developed by submergence.

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* Petroleum Development and Technology in 1925, 157. (A. I. M. E. Pamphlet No. 1570-G, issued with Mining and Metallurgy, April, 1926.)

† Associate professor of petroleum engineering, University of California.

‡ Petroleum engineering student, University of California.

5. For a 2-in. plunger pump, under varying conditions of length and speed of stroke, pumping a heavy, viscous oil, experimental evidence is offered which indicates that a submergence of from 100 to 230 ft. is necessary for 100 per cent. volumetric efficiency. With a lower submergence, the efficiency falls off rapidly, and for many tests, comparable with field conditions, volumetric efficiencies as low as 50 to 60 per cent. were obtained.

6. Proper pump submergence would, in many cases, require maintenance of fluid levels in the wells that would seriously reduce influx of oil from the reservoir sand. As a means of avoiding this difficulty, it is proposed that pumps should be placed in sumps drilled below the productive sands, unless this involves danger of water incursion.

7. To provide a basis for properly controlling both the fluid level and pump submergence in order to secure maximum efficiency in oil production, the use of a fluid-level indicator, attached to the pump and showing the pump submergence at all times, is recommended.

Mining Petroleum in France and Germany*

By George S. Rice† and John A. Davis,‡ Washington, D. C.

(New York Meeting, February, 1926)

THE PÉCHELBRONN OIL FIELD

The Péchelbronn oil field is located in the province of Alsace, in the Rhine Valley, about 30 miles north of Strasbourg. It is approximately 4 miles wide and 12 miles long, extending from the villages of Lobsann on the north to Wintershausen and Ohlungen near Haguenau on the south. (See Fig. 1.) The productive area includes about 44,000 hectares (107,680 acres). The name Péchelbronn signifies "pitch-spring" or "pitch-fountain."

The first exploratory galleries were driven in from the hillside in 1735; and in 1745 the first official concession was granted by the French government and the first pit sunk to a depth of about 30 ft. Extraction by drilled wells was begun in 1879, supplementing the mining operations, and during the next 10 years 200 wells were drilled to an average depth of 172 m. (568 ft.). The bringing in of two "gusher" wells, in 1882 and 1888, respectively, was followed by the abandonment of all underground work.

Underground mining, with a view to increasing the production of petroleum by recovering the oil left in the sands after drilling wells, was resumed in 1916. This mining operation started by sinking shaft No. 1 to a depth of 150 m. (495 ft.). The underground developments were pushed during 1917 and 1918 and the annual production of the Péchelbronn field was increased to about 50,000 metric tons or 350,000 bbl.

On the signing of the Versailles Treaty the province of Alsace, which had been taken by Germany as the result of the war of 1870, was returned to France, and the proprietary rights were taken over by the French Government. Concessions or leases, which embraced all the known oil-bearing territory of the field, were granted to the present operating company, Mines de Péchelbronn. This company followed both the drilling and the mining methods of the previous company.

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* Petroleum Development and Technology in 1925, 278. (A. I. M. E. Pamphlet No. 1570-G, issued with Mining and Metallurgy, April, 1926.) Published by permission of the Director, U. S. Bureau of Mines.

† Chief mining engineer, U. S. Bureau of Mines.

‡ Mining engineer, U. S. Bureau of Mines.

Geology of the Péchelbronn Field[1]

The petroleum basin of lower Alsace extends to the north of Strasbourg in the Tertiary Rhine plain. It lies between the Rhine River and the Vosges Mountains and not far from the latter. It is about 20 km. long and 7 km. wide, with its long axis running N. E. and S. W.

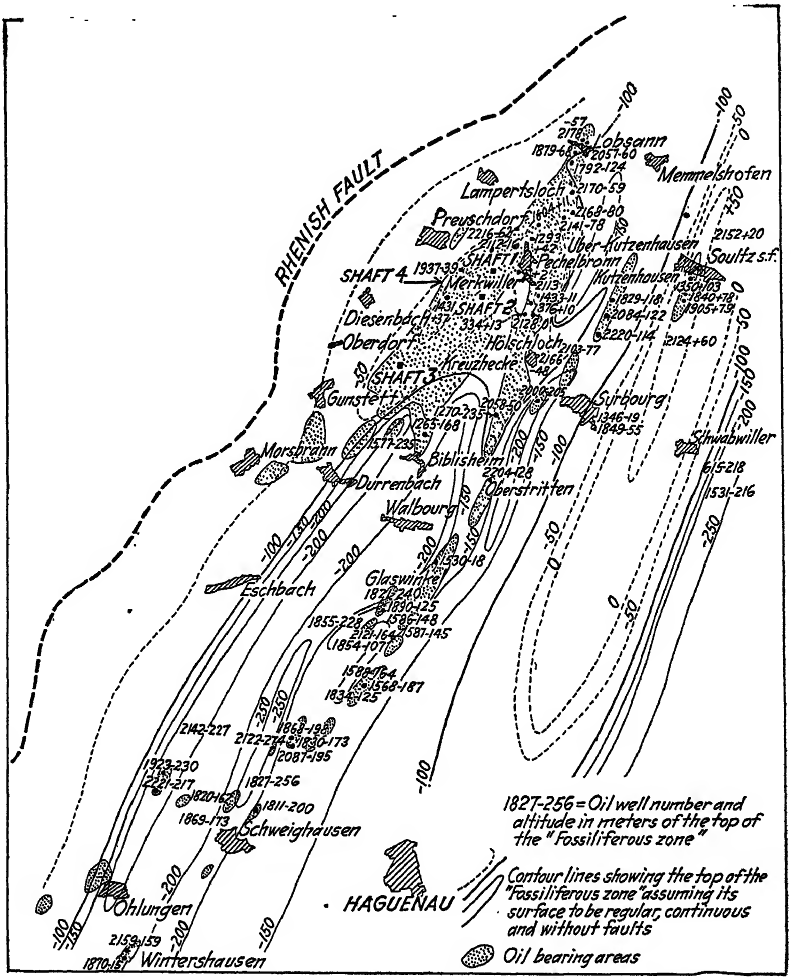


Fig. 1.—Map of Péchelbronn oil field. (Scale, 1:175,000.)

The western boundary of the Rhine plain is limited by the Vosges Mountains which consist of sandstone beds of Triassic age. (See Fig. 2.) These are cut off abruptly on their eastern edge by the Vosges fault, marking an uplift said to be 6000 m. To the east of this fault is a region

[1] M. Gignoux and C. Hoffmann: Le Bassin Pétrolifére de Péchelbronn. Serv. de la Carte Geol. d'Alsace et de Lorraine, Strasbourg (1920).

of smaller foot hills bordering the western edge of the Rhine valley, where the Mesozoic, if not concealed by loess, shows a highly-complicated faulted structure, a "veritable mosaic of Triassic, Liassic, Jurassic and even some Tertiary rocks." This faulted area is separated from the Rhine valley or Rhine fosse proper by a second large fault called the "Rhenish fault." In the Rhine Valley, which is formed over a great sunken monoclinal block, no outcrops of Mesozoic rocks are found. The petroleum basin lies to the east of the Rhenish fault and in places within a few hundred meters of it. The Rhine river running northward lies several kilometers to the east of the oil-producing sands.

The surface of the country is gently rolling and cultivated; the hills are well forested. Actual outcrops are said to be rare. The tops of the hills are covered with a considerable thickness of loess, but on the slopes the Oligocene marls come nearer to the surface.

A section of the Tertiary in the Péchelbronn region, constructed from drill-hole data, shows a thick series of marly formations which have been

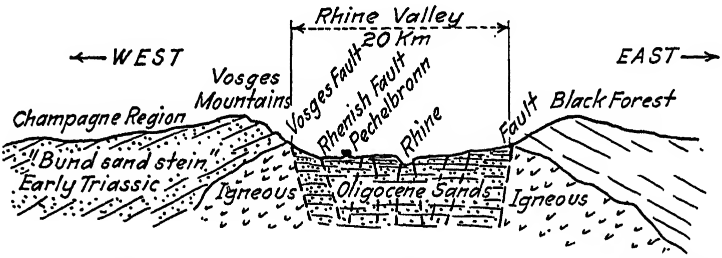


FIG. 2.—DIAGRAMMATIC GEOLOGIC SECTION OF PÉCHELBRONN OIL FIELD. (SKETCHED FROM MAP IN THE MINE OFFICE AT PÉCHELBRONN.)

very difficult to correlate. The beds differ slightly in color, texture, concretions, minerals, etc.—purely local distinctions. However, three fairly good reference horizons have been established: the so-called "Fish slates," the "Fossiliferous zone," and the "Red beds." Between these are thick series of marls and sandy lenses.

It is definitely determined that all the oil-bearing strata are of Oligocene age. The total thickness of the Oligocene is 1550 m. in this locality.

The Rhine plain, having sunk vertically, does not show the anticlinal and synclinal structure which usually plays such an important part in petroleum geology. The actual structure of the sand lenses has been obscured; but in general it is thought to be that of a much-faulted monocline.

Most of the numerous faults are post-Oligocene, as they affect nearly all the Oligocene formations. The faulting is roughly parallel and trends in a general northeasterly direction. The faults range in importance from mere slips of 3 to 5 m. throw, caused by shrinkage and contraction of the clay beds, to larger faults which cut up the field in vertical slices,

generally parallel to the Rhine. The productive zones seem to be long and narrow, following the orientation of the sand lenses which have their longer axes also parallel to the general northeasterly—southwesterly trend. A slight enrichment of oil has been noticed near the faults, which has been attributed to the sealing action of the latter and does not indicate, as was formerly believed, that the faults themselves had served as channels of access for the oil to the sand lenses.

The elongated lenticular masses found at depths of 50 to 60 m., as well as the deeper flatter beds, are arranged parallel to the main faults, which form the escarpment of the Vosges Mountains and of the Black Forest hills lying east of the Rhine River.

Origin of the Oil in the Péchelbronn Basin

Evidence from underground workings indicates that the oil sands were deposited very rapidly and immediately covered and sealed off by clayey ooze. Hence the petroleum itself must have been formed at some other place and transported with its enveloping sand to its present position. The French geologists do not believe that the oil gained access to the sand lenses in their present position by means of the faults. The absence of water under hydrostatic head would seem to support their conclusion.

Nature of the Oil of Péchelbronn

Three distinct kinds of oil are found in the Péchelbronn field: A thick asphalt oil, with an average density of 0.970, encountered at depths of 50 to 60 m.; a heavy crude oil with an average density of 0.945, occurring at depths of 70 to 100 m.; and a lighter oil, with an average density of 0.880, found in the lower strata.[2] The two heavier oils are found in narrow elongated lenticular deposits of sand and are the oils which were exploited from 1735 to 1888 by means of shallow pits and galleries.

The lighter oil is more gaseous. It is found in thinner and flatter sand lenses which are from 2.5 to 3.0 m. thick, 200 m. wide, and up to 1 km. long. They lie at a depth of 150 to 600 m. The important difference between these oils from the viewpoint of mining development lies in the fact that the heavy shallow oils have very little gas, and the podlike lenses of sand containing them could be worked by the early miners with open lights without grave danger.

Development of Péchelbronn Oil Sands by Drilling

In 1919, the production from 500 wells then in operation was 30,000 m. tons (about 210,000 bbl.), a small portion coming from flowing wells, but most from pumping wells. M. de Chambrier, to show the relation

[2] Paul de Chambrier: Les Gisements de Pétrole d'Alsace. *Bull.* Soc. d'Encouragement pour l'Industrie Nationale, Paris (1920), Jan.–Feb., 15.

between flowing and pumping, states[3] that the 45 wells at Péchelbronn on March 31, 1902, had produced 46,071 m. tons of oil by flowing, and 125,945 tons by pumping; or in the proportion of 26.6 to 73.4. Since then the proportion of oil by flowing has gradually dropped so that it was insignificant in August, 1922, although there were 500 wells in operation.

In all, over 1000 wells had been drilled by 1923, including those from a shallow depth to those with a maximum depth of 500 m. (1650 ft.). The average depth was 300 to 350 m. (990 to 1150 ft.).

The following table shows the production of crude oil in Alsace, all of it from the Péchelbronn field:

Production of Crude Oil in Alsace[4]

Years	Total Production		Yearly Average	
	Metric Tons	Barrels*	Metric Tons	Barrels*
1735 to 1811	3,000	21,000	39	275
1812 to 1866	4,959	34,500	90	630
1867 to 1881	9,207	64,500	614	4,300
1882 to 1888	33,771	236,000	4,824	33,700
1889 to 1905	303,148	2,120,000	17,832	122,000
1906 to 1917	460,890	3,230,000	38,401	269,000
1918 and 1919	98,449	689,000	49,225	344,000
1920	54,910	385,000	54,910	385,000
Total	968,334	6,780,000		

* U. S. barrels.
Surface pits and underground mining from 1735 to 1888.
Wells, exclusively, operated from 1889 to 1916.
Mining and wells from 1917 to 1920.

Well Drilling Methods

The drilling was done by the ordinary churn drill with rope and walking beam, the drilling outfit being small as compared with the drills used in the United States. Wells usually flow for only a few months, but the one producing the largest total flow[5] struck the oil in November, 1884, and continued to flow for 11 years, producing 10,791 tons (75,000 bbl.). Subsequently by pumping it had produced, up to 1918, an additional 8876 tons (62,000 bbl.).

The pumping was done by a deep-well pump connected to an I-beam used as a walking beam and this was driven by a small electric motor of 2 or 3 hp. The electric power was distributed from a central power station and the oil pumped to a refinery.

[3] Paul de Chambrier: La Source de Pétrole Jaillissante de Péchelbronn. *Bull. Soc. d'Encouragement pour l'Industrie Nationale* (1920) July–Aug., 5.

[4] *Pet. Times* (1921) 211.

[5] Paul de Chambrier: *Op. cit.*

In 1923, the total length of pipe lines from the wells to the refineries was 127 km. (76 miles) and the length of electric power lines was 74 km. (44 miles).

CHARACTER OF OIL FROM WELLS AT PÉCHELBRONN[6]

	Average	Flowing Well
Density	0.888	0.874
Viscosity at 50° C.	2.2	2.3
Freezing point	−5° C.	−20° C.
FRACTIONAL DISTILLATION[6]		
Percentage distilled at 150° C.	5.90	1.30
Percentage distilled 150° C. to 300° C.	27.80	24.09
Percentage distilled	66.30	74.61
	100.00	100.00

REFINING OF THE OIL

The following description of the refining methods at Péchelbronn has been taken from the report of Chester W. Davis, U. S. Consul at Strasbourg, France.[7] The refining operations include the following:

Treatment of the Crude Oil by Decantation and Drying

The oil as it emerges from the earth contains varying portions of salt and muddy water, which must be separated by decantation and by heating under pressure in boilers with closed hearths. This operation is known as "drying the crude oil."

During the drying a distillation occurs and the gas and volatile oils separate little by little. There finally remains in the "alembic" dry crude oil which still contains some kerosene and the lubricating oils. The crude gas and petroleum are redistilled and rectified or treated chemically to separate all the asphaltic contents which might tend to lessen their quality.

Distillation in Vacuum

The dried crude oil is concentrated in "alembics" of 30 tons capacity each. In order to prevent the decomposition of the oils under the action of the heat, and to increase their condensation, the distillation is carried on under reduced pressure. The use of the vacuum and continuous distillation are said to result in a more rapid distillation under moderate temperatures (not exceeding 330° C.), in an economy in fuel and, above all, in the production of lubricating oil of a greater viscosity and consequent greater value.

[6] Paul de Chambrier: *Op. cit.*, 6.
[7] Chester W. Davis: Consular Report (Oct. 23, 1924).

Separation of the Volatile Products

The gas dissolved in the petroleum escapes under the influence of the heat and is passed through a gasometer. It is compressed and then liquified, and an extra volatile gasoline is drawn off, the initial boiling point of which is less than 15° below zero (Centigrade).

Removal of Paraffin

Improved machinery completely removes the paraffin from the lubricating oils after which they are redistilled in order to produce spindle oil, machine oils, cylinder oil and superheating oils.

Chemical Treatment

The oils are submitted to a chemical treatment in order to give a transparent appearance. This treatment consists in causing the resinous content of the oil to be absorbed by concentrated sulfuric acid. The solution is then neutralized by caustic soda.

Treatment of Paraffin

The paraffin gathered by the filter presses is refiltered while warm and the oil removed. It is then reheated, although not to the melting point, to remove the last traces of oil. After being treated chemically, it is molded into flat white cakes for the market.

Disposal of "Bray" and Petroleum Coke

The "bray," which is the first residuum of the distillation of lubricating oils, is marketed in its original form or is still further distilled. The petroleum coke remaining after this final process is sold for the manufacture of electrodes.

REFINERY PRODUCTS AT PÉCHELBRONN IN 1923[8]

Product	Production in 1923, Metric Tons	Increase or Decrease Compared with 1922, Metric Tons
Gasoline	3,848	+ 771
Kerosene	16,866	−1,057
Refined oils	16,331	+1,652
Non-refined oils	979	− 70
Grease and fuel oil	16,557	−6,499
Paraffin and paraffin products	1,702	+ 96
Bray	9,135	+ 667
Coke	102	+ 92

[8] Chester W. Davis, *Op. cit.*

The capacity of the refineries is small as compared with those in the United States. (See Fig. 3.) The following flow sheet indicates the products and objectives in refining. Eight commercial products are indicated by number.

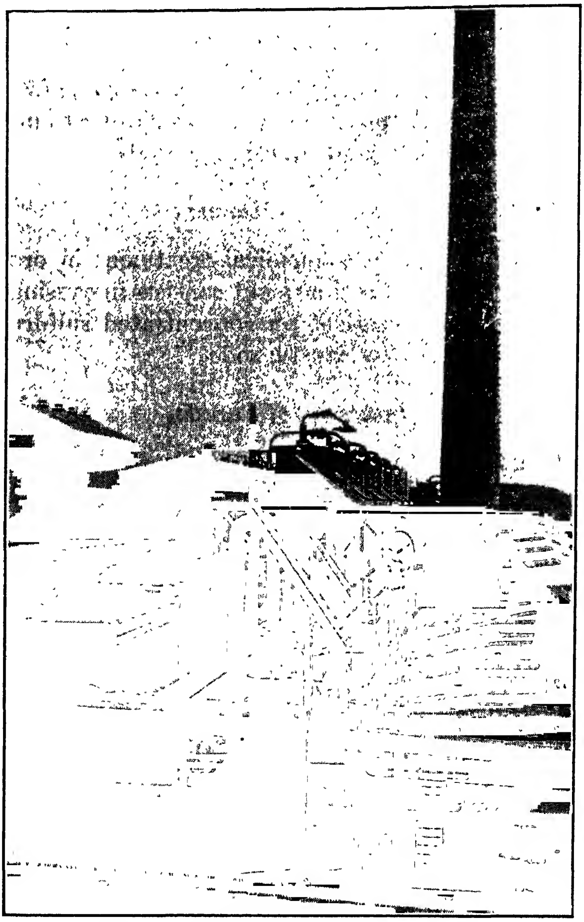


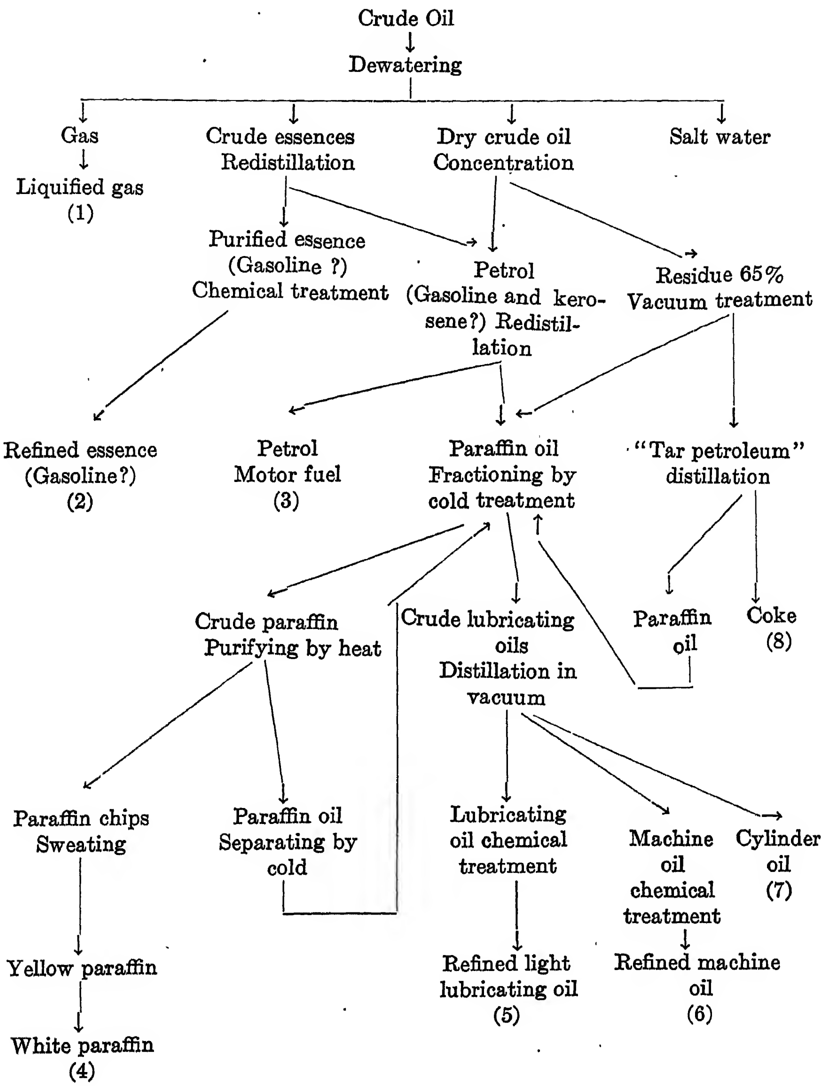
FIG. 3.—DISTILLATION APPARATUS.

CONSIDERATIONS LEADING TO MINING THE OIL LANDS

The amount of oil retained by the sands after drilling opeations is influenced by the state of compression of the sand. De Chambrier reports[9] that when washed sand is compressed to 65 or 70 per cent. of the

[9] Paul de Chambrier: Exploitation du Pétrole par Puits et Galeries (1921), 19.

FLOW SHEET OF PÉCHELBRONN REFINERY[10]



10 Paul de Chambrier: Les Mines et la Raffinerie de Péchelbronn (given at The nstitute of Chemistry of the University of Strasbourg, June 1, 1920), 28 pp.

original volume it will absorb less oil than free sand, but the decrease in absorbtive power is proportional to the decrease in volume. The amount of oil remaining in the sand after draining is proportionately but little less in the compressed sand than for the free.

From experiments made in 1914 and 1915 the following results were obtained of the quantity of petroleum which may be held by sands.

TESTS OF ABSORPTION OF OIL BY SAND[11]

	Percentage by Volume	
	Free Sand, Per Cent.	Compressed Sand, Per Cent.
Absorptive power	40	27
Oil draining off	13	9
Oil remining in sand	27	18
	Percentage by Weight	
Absorptive power	22.7	12.7
Oil draining off	7.4	4.2
Oil remaining in sand	15.3	8.5

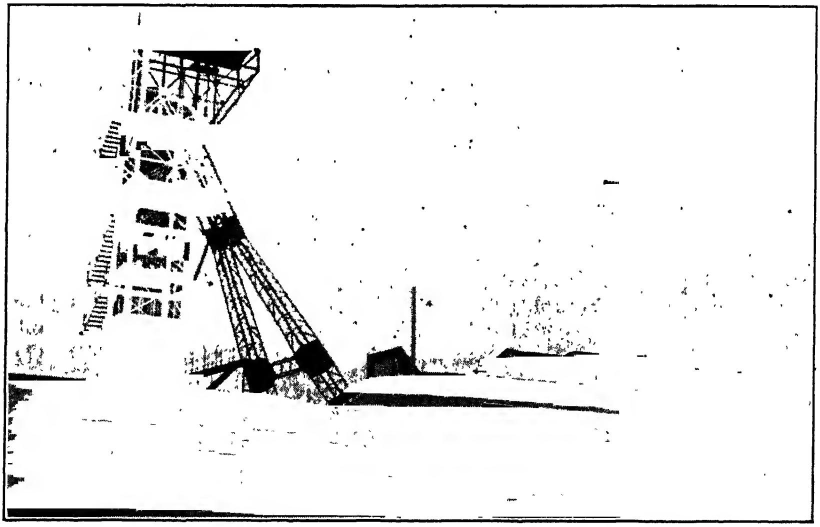


FIG. 4.—HEADFRAME AND SHAFT BUILDINGS AT CLEMENCEAU PIT.

The French technologists have finally arrived at the conclusion that for every barrel of oil obtained by wells, five other barrels remain in the sand unobtainable by pumping operations. Furthermore, it is stated by de Chambrier[12] that, based upon data obtained from experiments, 200 l. of oil per cu. m. of sand must remain held in the sands by capillarity or by surface tension, after production from wells has ceased. Part of this can

[11] *Op. cit.*, 20.

[12] *Op. cit.*, 22.

be obtained by draining, but part cannot be recovered except by washing the sand with hot water.[12a]

After a lapse of 28 years, when it was found that production in the field from drilling had reached its zenith and laboratory tests had indicated the large loss of oil by the well drilling method, it was decided in 1916 to return to underground mining in those areas where the limit of oil and gas production by wells had been reached. Accordingly the operating company—the Deutsche Erdöl-Aktiengesellschaft ("Dea") under the pressure of the greatly increased German Government needs for oils and greases caused by the war, began the sinking of Pit No. 1 (now called the Clemenceau Pit). It was completed to a depth of 200 m. (660 ft.) in April, 1917. (See Fig. 4.)

Results of Underground Mining at Péchelbronn

As a result of the readoption of underground mining at Péchelbronn, the production of the Alsatian oil field which had been declining during the war when most needed by the Germans, was promptly raised and has since been increasing with extended mining developments. Data for the following table has been collected from several sources.

Oil Production at Péchelbronn by Years

	From Wells, Metric Tons	From Mines, Metric Tons	Total, Metric Tons
1900	22,576	0	22,596
1910	33,493	0	33,493
1912	47,176	0	47,176
1913	49,584	0	49,584
1914	49,055	0	49,055
1915	43,176	0	43,176
1916	41,579	0	41,579
1917	39,124	7,787	46,911
1918	32,019	19,174	51,193
1919	30,300	16,955	47,255
1920	42,025	12,885	54,910
1921	43,825	11,750	55,575
1922	45,150	24,960	77,110
1923	33,226	37,469	70,695
1924 (estimate)	30,000	45,000	75,000

[12a] As an offset to the more enthusiastic estimates of oil held by sands after drainage, an interesting and curious but exceptional case is mentioned by de Chambrier. (Exploitation dur Pétrole par Puits et Galeries, p. 69.) He says that a place near an old well and also located in a part of the sand bed which has been completely drained of its oil by galleries, a mass of dry and friable sand was encountered which contained only 0.7 to 1.4 per cent. of oil. It was, however, saturated with gas. "It seems therefore that the gas has the power to free the sand almost completely from its oil. There was no water in this bed and the content of clay was low, 5 per cent."

In the area now being mined from the No. 1 Pit (Clemenceau) there had been four pumping wells which produced from 1908 to 1917, a total of 21,000 tons (147,000 bbl. of oil). From March, 1917, to Sept. 1, 1920, the oil produced from this same area by drainage into the mine galleries reached a total of 48,400 tons (338,800 bbl.) or about 2½ times as much as by the four wells. Further it is known that there is a large amount of oil which has not drained into the galleries and still remains in the unmined sands. Whether it is commercially practicable to obtain this oil was unsolved, at least up to 1923, but the mine management at that time thought it could be done although the proposed method of mining was not described.

De Chambrier estimates[13] that a ton (2205 lb.) of the Péchelbronn oil sand contains 120 kg. (264 lb.) of crude oil, or 12 per cent. by weight, from which the recovery should be somewhat as follows:

Recovery from Péchelbronn Oil Sand

		Per Cent. of Total
From wells	20 kg. (44 lb.)	16.7
From drainage into mine galleries	52 kg. (114 lb.)	43.3
Remaining underground	48 kg. (106 lb.)	40.0
	120 kg. (264 lb.)	100.0

On the basis of volume, the yield at Péchelbronn from both wells and galleries is equivalent to 20 gal. per cu. yd. with 13 gal. per cu. yd. remaining if the sand is left unmined.

Physical Character of the Deposits as Observed in the Mine

The oil occurs in various sand beds ranging in thickness from 2 to 2.5 m. (6½ to 8 ft.), and interbedded with limey marls. The strata are cut by faults into irregular long narrow blocks. The dip is irregular but in the Clemenceau mine it averages 5° to 6° in an easterly direction. At the faces visited by Rice and Litchfield in 1923, there seemed to be a concentration of oil along the marl floor as the lower portions of the sand beds were seen to be thoroughly saturated with oil. The upper part of the sand bed seemed to have much less oil, but some drainage from it undoubtedly takes place during the advance of the heading faces. The marls of the roof and floor were apparently impervious to oil.

The sands themselves where examined, were of medium grain, quite compact and friable. A loose chunk of the material was easily broken

[13] *Op. cit.*, 72.

by the fingers and yet was singularly tough in place. It was difficult to drive the point of a hand pick directly into the face and it was noted that the point of a pneumatic pick had to be held diagonally against the face for several seconds before a piece would slab off. After the first piece had been loosened it was easier to chip off adjacent pieces. No cleavage and very few bedding planes were seen. At one spot a distinct cross bedding of the sand was noticed. There were a great many slip planes and thin veinlets or seamlets of the bluish marl.

The sands vary greatly in texture from place to place and also in clay content. The most impure and clayey sands were stated to be the richest in oil.

The marl, where observed, was a homogeneous grey limey material which as already stated was apparently impervious to the oil. The contact between the sand and marl was very sharp. Gypsum and anhydrite veinlets were noted in specimens of the marl in the geological collection. The marl was stronger as a roof material than the sand, although it evidently moves under pressure and is more friable; that is it can be "picked" off more easily in a freshly exposed face.

Pyrite nodules were said to be occasionally found. Pieces of wood impregnated with petroleum which had been encountered in the sands were shown in a mineral collection.

Character of Roof and Floor

In the headings visited the marl roof was "heavy," and the closely placed timber sets along the galleries showed considerable bending, although de Chambrier states[14] that the ground cannot be classed as "heavy," as the marl is strong and hangs well. Both the roof and the foot wall or underlying stratum were usually marl. The gallery floor in the levels was in marl on the raise side and sand on the dip side. In the inclines the floor was sand. It had to be covered with planking, on account of softness.

Absence of Water in Strata

In the writers' opinion the absence of water under hydrostatic head in the sands is an important factor in making the method of mining commercially practicable. No water was observed anywhere and in fact none is said to be found in the workings. There is said to be some salt water in a few of the wells, however, and a trifling amount is mined with the oil, but never under pressure. All of the oil sands in the galleries were remarkably free from it.

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[14] *Op. cit.*, 39.

Gas Pressure in Mine Workings

The absence of any appreciable gas pressure left in the sands being mined is another factor which contributes greatly to the practicability of the method of underground mining. At the beginning of the 19th century before well drilling had drained most of the gas from the area now being mined some gas pressure was encountered in the old pits.

One drill hole was being put down into the sands at the time of the visit from the 150-m. level (150 m. below sea level) to relieve any gas pressure and it was admitted that in new areas a little gas is sometimes found which is relieved in this way. But as a rule the present workings do not meet with any serious quantities of gas under pressure.

General Layout of the Mine

With the resumption of mining in 1916 it was decided to attack the oil sand strata directly with galleries and inclines, taking adequate

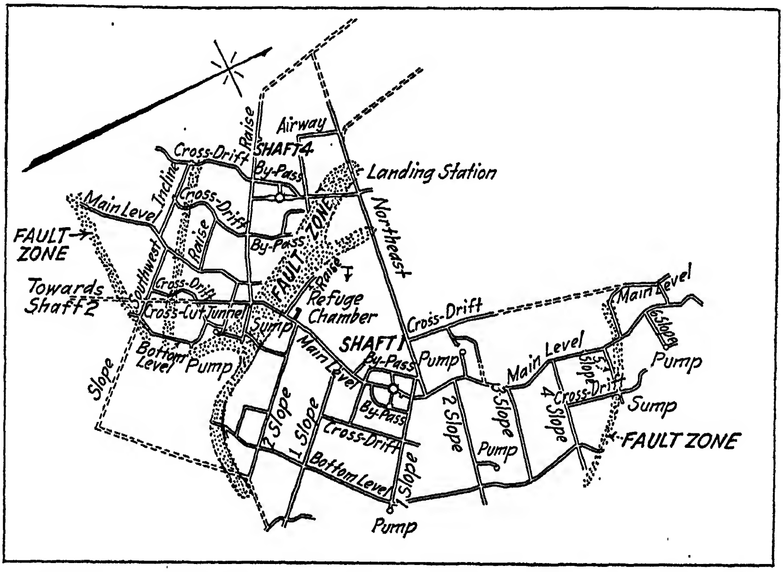

FIG. 5.—MAP OF CLEMENCEAU WORKINGS. (SCALE, 1:7000.) SHAFTS 1 AND 4 ARE THE SMALL CIRCLES ON THE DRIFTS BETWEEN THE BY-PASSES.

precautions for ventilation and to permit easy escape by the men, rather than to employ the old method which was abandoned in 1888, and which consisted in driving the main headings in the marl and clay above the oil sands with short drainage cross-cuts at intervals down to the oil sands.

Four shafts serving two mines had been sunk prior to August, 1923. Pits No. 1 (called the Clemenceau Pit) and No. 4 served the first mine. Pit No. 1 was the hoisting shaft, and also the "down cast shaft" for the

ventilating current which, after circulating through the mine, "returned" through Pit No. 4, the upcast shaft.

Pits 2 and 3 served a new mine. The first mine had a daily output of 100 cubic meters or about 88 m. tons of oil and the second, or new mine, an output of 28 cu. m. or about 25 tons of oil.

From these shafts, level and inclined drainage galleries in the sand bed are laid out roughly at right angles to each other, insofar as the changing strike and dip permits. The main levels follow the contours of sand beds and therefore are very irregular in direction. (See Fig. 5.)

The practice was not to mine and hoist sand other than obtained in advancing the galleries. The sand coming from the headings was stacked in piles on the surface awaiting the erection of a washing or treating plant. Meantime a certain amount of oil seeped from the piles and was collected by ditches.

It had been determined that it was sufficient for oil drainage purposes to put the main galleries from 50 to 100 m. (160 to 350 ft.) apart with inclines connecting them at 50-m. (160-ft.) intervals.

De Chambrier[15] classifies the galleries as follows:

1. Main development and ventilating galleries; horizontal.
2. Inclines following direction of greatest dip (dip varies from 5° to 6°).
3. Cross-cuts for special purposes.

The main galleries visited were 2 m. by 3 m. wide (6.5 by 10 ft.) measured inside the timbering. The temporary inclines and cross-cuts were somewhat narrower.

Timbering and Lining of Galleries

The gallery or drift timbering (there were no chambers or rooms) was unusual only in the amount required, for the timber was continuous and close set, with continuous lagging. Ordinary sets of round timber with notched cap or collar were used, the legs being rather steeply sloped. The timbers were somewhat lighter than those in most metal mines of the United States but were about the same size as those in the average coal mine, *i. e.* 6 to 8 in. in diameter. Timbering had to be put in as fast as the face advanced; in some instances forepoling was required.

The timber sets at the face were 1 m. (3.3 ft.) from center to center. Spacing blocks were employed. Subsequently, intermediate sets were put in and in some places sets were placed "skin to skin." In all cases tight lagging of small poles was employed both on the top and sides. The posts were on blocks or on cross-sills. Floor planking was spiked to the sills, except over the oil ditch when that was placed centrally, which was the more recent practice.

[15] *Op. cit.*, 40.

Near the shafts the galleries were walled with brick, and steel rails were laid across the tops of the walls. The rails were spaced closely enough for cement blocks to be set between them resting on the flanges. Throughout the rest of the mine, however, wood timbering, which showed considerable bending from the gradual squeeze, was generally employed.

De Chambrier in his paper[16] proposed as a fire protective measure that all wood timbering be done away with in oil sand mining and that steel or iron "sets" be substituted with employment of small pipes or plates for lagging, but this substitution had not yet been used in the galleries visited in August, 1923.

"Guniting" the walls of the headings with a thin coat of sand cement, employing the American "cement gun" for the purpose, had been tried but it was stated the coating did not stick to the marl which swells when wetted. The management was continuing to experiment with different methods of application. At the time of the visit, the cement gun was being used in filling a cement-sand mixture behind forms to make a massive fire proof lining for a gallery in the vicinity of an oil sump.

Gutters and Ditches for Draining the Oil

When mining was first resumed in 1916 the floor of the heading was cut entirely in the marl foot wall and ditches for the oil were made along

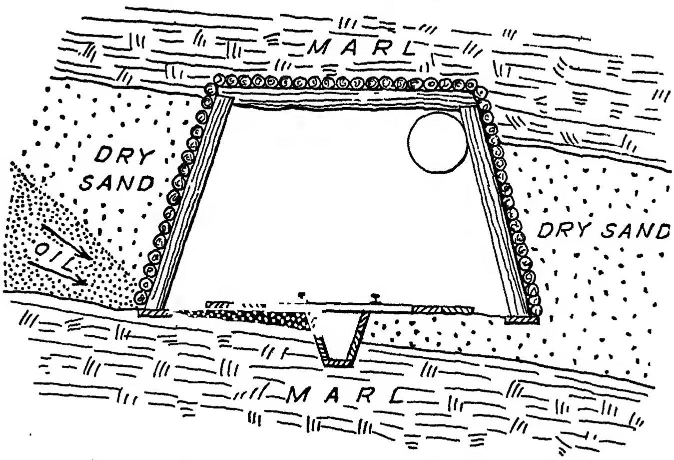

Fig. 6.—Section of gallery at Péchelbronn.

each side in the marl. With this system the oil draining from the high side tended to flow over the floor, although iron sheets were used for the bottom lagging, and was objectionable in transportation and presented a fire hazard. Later the practice was changed and the floor of the heading was kept in the sand whenever possible, letting a central ditch cut down

[16] *Op. cit.*, 94.

into the marl under the floor. Usually the ditch was lined with sheet iron, plank or concrete and was also covered with plate for fire protection. (See Fig. 6.)

By driving the bottom of the levels above the marl floor of the sand bed the top of the "seepage" zone was a minimum distance above the floor of the gallery, hence there was a minimum of gallery wall exuding oil and a minimum of oil exposed. The new arrangement could not always be carried out, however, in levels where the dip was steep.

The oil sinks under the floor planks at the sides of the gallery or seeps between the planks and works its way into the ditch. The gutters in the inclines carried the oil to the ditches in the level below. The oil draining into the inclines below the lowest level was collected in a temporary sump and pumped up to the level. The levels have a slight grade toward the permanent sumps. Finally all the oil was discharged into a main central sump.

Sometimes to assist drainage of oil from the sands or local pockets in the roof, holes were drilled from the galleries on the raise side. This practice was not usual at Péchelbronn but was often practiced at the Wietze mine in Hanover.

Oil Sumps and Pumps

At the bottom of inclines and at other low points small sumps were made. These local sumps were brick-lined, and the oil was pumped from them by compressed air pumps to the main sump in a brick-lined room offset from the "main bottom" tunnel. It was pumped to the surface through a bore hole by a deep-well pump, the head of which was on the surface. Thus it will be noted that the main sump and the pipe line were not located in one of the shafts, as was the case before a serious fire occurred in 1919. The only entrance to the sump chamber had an iron fire door ready for closing.

Drainage of the Oil, Theoretical Considerations

De Chambrier has treated[17] the general subject of oil seepage and circulation exhaustively. The circulation of the oil was very slow in the sands in place. When a cross-section of the bed was exposed by a heading, it was noticed that in a bed 2.5 m. thick the oil was generally drained from the lower 50 to 70 cm. (20 to 28 in.) of the bed, with the greater volume draining from the very lowest part of the bed.

When pumping in an oil field, hitherto tapped only by wells, ceases to be effective for production de Chambrier[18] believes that the remainder of the oil between the sand grains exists in the form of small globules separated by bubbles of gas whose surface tension prevents the flow of the viscous

[17] *Op. cit.*, 51.

[18] *Op. cit.*, 64.

liquid. Under reduced pressure the gas has no longer the power to migrate through the minute passages in the bed and the oil merely by its own weight cannot overcome the frictional resistance in the capillary openings when these are obstructed by the gas bubbles. Hence a large amount of oil must remain in the sand unobtainable by drilling operations. If the sand beds are now exposed and opened by galleries the equilibrium is broken, the gas can escape more easily and the oil no longer having this obstacle to overcome can sink down through the sands of its own weight and join the ever-thickening sheet of oil moving along the floor of the bed toward the galleries.[18a]

Methods of Mining at Face

Owing to the possibility of striking sparks that would ignite the highly inflammable gas when that is given off, the employment of hand picks which strike a glancing blow is discountenanced. Instead of this, the pneumatic hammer pick is used with the point held steadily against the oil-cemented sand and at an angle with the surface to slab it off. No explosives are used and there is no occasion for their use.

The marls and clays of the roof and floor are dry of oil. They were easier to pick than the sand but were tougher. Nevertheless the squeezing of the roof (the pressure is largely vertical) begins immediately at the face, so timbering was put up as fast as the excavation proceeded. In fact, the practice was to make cuts into either side of the face and across the top and put in a timber set recessed in the cuts as quickly as possible, and also lagging was put in before the central portion of the face was excavated.

Transportation and Hoisting of the Sand

The sand at the face was loaded into small, tight steel-plate cars, holding about ½ to ¾ ton of sand. The cars were hauled to the hoisting shaft by horses or mules, hoisted to the surface, and dumped on a storage pile.

Shaft and Equipment

The No. 1 Pit (Clemenceau) is a circular shaft about 4 m. (13 ft.) in diameter, has two compartments, and was 300 m. (660 ft.) deep. The upper part is lined with brick, the lower part with massive concrete. The two hoisting cages were in balance. They had rail guides and safety catches; the use of the latter was unusual in Europe. The cages and head frame were small but of steel construction. The hoisting engine was steam-driven of the usual type for small capacity hoisting.

[18a] An experiment was made of forcing gas into a drill hole. It was found that a pressure of 360 mm. (5 lb. per sq. in.) was required to force gas through 50 cm. (20 in.) of sand. De Chambrier, therefore, concludes it would take an enormous pressure to force petroleum through hundreds of feet of sand.

Washing the Oil Sand

From 1735 to 1867 the oil refinery at Péchelbronn treated the bituminous sands extracted from the mine with boiling water. In 1918 a new washery was installed; the sand was agitated by paddles in large tanks filled with water at 80° C., heated by exhaust steam. The system did not give the desired results. One difficulty was the rapid wearing away of the paddles. No washing of the sands was being practiced in 1923. It was stated that some new system of washing would be used for the sands in the storage piles.

Mine Gases and Physiological Effects

The mine was considered very gaseous, and there were other gases in addition to methane which are exceedingly sensitive to ignition. Methane has no physiological effect but certain components in the mine gas mixtures had a sweetish taste and odor and had a toxic effect on the men. It was stated that before the gas coming in at the face attained dangerous proportions the men began to have headache and soreness of throat, which automatically gave warning. When this occurred the men were immediately withdrawn from the respective heading. A content of only 3 to 4 per cent. of the gases in air would make an explosive mixture. Very much smaller quantities rendered air unfit for breathing. According to de Chambrier samples of the pure gas taken at the faces give the following average analysis:[19]

Constituents of Gas from the Péchelbronn Oil Sands

	Per Cent.
Methane	80.0
Saturated hydrocarbons	16.0
Olefines (unsaturated hydrocarbons)	.5
CO_2	.3
Nitrogen	3.0
Oxygen	0.2
CO	0.0
	100.0

This analysis does not indicate definite physiological effects would result, if the gas is in small proportions in the air. It is interesting to compare the above analysis with analyses given by de Chambrier for gases coming from four flowing wells.[20]

[19] *Op. cit.*, 46.
[20] Paul de Chambrier: La Source Pétrole Jaillissante de Péchelbronn (1920), 7.

ANALYSES OF GASES FROM FLOWING WELLS

Number of wells	220	1261	1535	2183
Date of analysis	1912	1912	1913	1920
Methane	76.5	96.2	97.5	66.0
Ethane and saturated hydrocarbons	16.2			24.4
Olefines (unsaturated hydrocarbons)	0.4	0.6	.....	.....
Carbon dioxide	.....	0.2	0.4	0.5
Oxygen	0.5	0.5	.....	1.0
Nitrogen	6.4	2.5	2.1	7.7
H_2S	.....	.....	.....	0.4
	100.0	100.0	100.0	100.0

It was stated only rarely were traces of carbon monoxide and hydrogen found in mine gases, but this fourth sample indicates a dangerous amount of hydrogen sulfide.[20a]

De Chambrier states[21] that a spark produced by the blow of a pick on a hard rock does not attain a high enough temperature to ignite methane, except in the presence of hydrogen or certain hydrocarbons, but the latter are much more sensitive than fire damp and can be ignited by this means.

Because of danger of igniting gas, no electricity was used in the mine except in storage battery miner's hand lamps. These lights had an illuminating power rated at 1.5 cp. and burned for 10 hr. on a single charge. De Chambrier says,[22] however, that ordinary incandescent electric lights would be satisfactory in main galleries but not at the face. Flame safety lamps were not used. It was understood this was because they were thought unsafe in the gas mixtures encountered. The use of explosives in the mine is forbidden, but in any case they are not needed.

VENTILATION OF MINE

What was considered to be a very elaborate system of ventilation was maintained. The hoisting shaft (Pit No. 1) was the downcast and the air shaft (Pit No. 4) was the upcast, at the top of which was an exhausting fan. The air was coursed through the galleries by stoppings and doors at appropriate places. The ventilating current had three separate splits. The principal ventilating doors were installed in triplicate and made of steel. As the headings advance beyond the last cross cut, small auxiliary fans were placed at the outby side of the fire stoppings with 50-cm. (20-in.) diameter metal air pipes through the

[20a] Hydrogen sulfide is highly poisonous. Dr. R. R. Sayers, chief surgeon of the Bureau of Mines, states that 0.06 to 0.1 per cent. is enough to cause serious symptoms within a few minutes. Lower concentrations cause headache, dullness, dizziness, pain in the eyes followed by conjunctivitis.

[21] Paul de Chambrier: Exploitation du Pétrole par Puits et Galeries (1921), 47.

[22] *Op. cit.*, 55.

stoppings to ventilate the faces. These fans were run by compressed air engines. Also a very compact and apparently efficient small Rateau turbine blower was observed, which had the same diameter as the ventilating pipe. Due to this system of "auxiliary fans" the air seemed to have more velocity at the faces than in the main airways.

The following table gives an average analysis of the air in the main upcast.

Analysis of Air in the Main Upcast at Péchelbronn

	Per Cent.
Oxygen	20.0
Nitrogen	79.5
Carbon dioxide	0.2
Other gases, methane, oil gas, etc	0.3

The ventilation according to de Chambrier[23] must be sufficient to keep the percentage of hydrocarbon gases below 0.2 per cent. Six cubic meters or 210 cu. ft. of air per man per min. was the amount fixed on. The amounts specified for gassy coal mines in different states of this country vary from 150 to 200 cu. ft. per man per min. Samples of gas were taken daily in the returns at the working faces, and evidently the ventilation was well administered.

Fire Hazard and Precautions

As the oil sand gases are exceedingly inflammable, the mine officials stated the production of any sparks is to be avoided. If the gases are ignited, a disastrous oil fire is sure to follow. It was for this reason the use of air-hammer picks is required instead of hand picks, because the latter were apt to give a sliding, glancing, blow and produce sparks.

The accumulation of oil in the working places is to be avoided, on account of the fire risk. Sheet-iron sheathing between the walls of the galleries and the timbering was considered a commendable thing, also steel sheets over the ditches. So far as practicable the use of combustible material should be done away with, but this measure had not been carried out except in the vicinity of the shafts and main sumps. Also as a precaution oil is no longer pumped out through pipe lines in the shaft but through bore holes, put down to the main sumps for the purpose.

Fire extinguishers were kept in all working places. At the faces visited there were two carbon-dioxide extinguishers of the Babcock type. The men had directions to use these first in case of fire; if it could not be gotten under control by this means, they were instructed to get out from the face and close the fire-doors.

Within 50 m. (165 ft.) of each face a fire door is required consisting of an iron door hung in a tight brick frame. (See Fig. 7.) A ventilating

[23] *Op. cit.*, 48.

pipe passed through the stopping. In case of fire the door is closed, which automatically dropped a steel plate slide across the ventilating pipe and thus closed all openings. As the heading advances the fire door is moved forward from time to time. These doors were also regarded as explosion stoppings but did not look strong enough for such purpose.

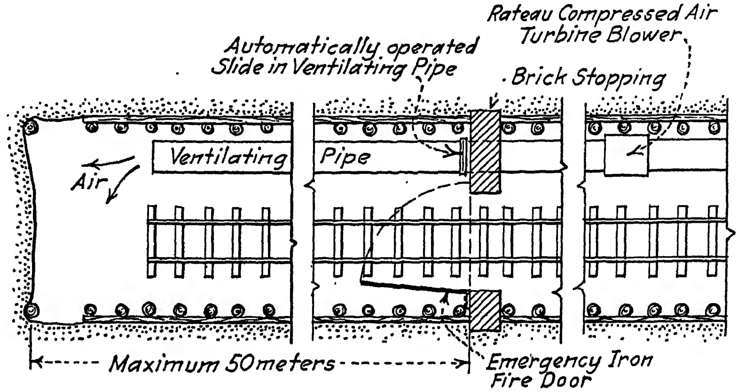

FIG. 7.—PLAN OF A HEADING AT PÉCHELBRONN OIL-SAND MINE, SHOWING ARRANGEMENTS FOR VENTILATION AND EMERGENCY FIRE DOOR.

To isolate large areas of the mine heavier doors of the swinging type were used. (See Fig. 8.) They were mounted in heavy concrete frames. They looked very strong and were said to be explosion-proof. It was

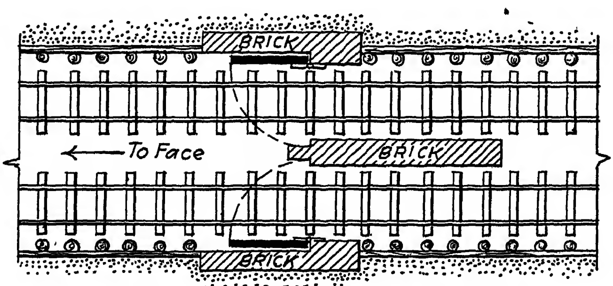

FIG. 8.—PLAN OF "EXPLOSION DOORS" EMPLOYED AT PÉCHELBRONN TO ISOLATE PANELS OF MINE.

stated by mine officials that sand bags are provided behind doors for the rapid erection of additional barriers.

Another feature which has been described by de Chambrier[24] was the refuge chamber which was isolated from the rest of the mine by two iron

[24] *Op. cit.*, 56.

doors. It was kept furnished with food and water and rescue apparatus. There was telephonic communication between it and the surface. It was independently ventilated by two pipes to the surface, one for intake and one for exhaust. The chamber was for refuge in case the escape to both shafts were cut off.

Explosions, Fires and Accidents in the Past

The earliest accident at Péchelbronn of which there is any record is described in the *Journal des Savants* of 1759: a gas explosion (they were at that time working with open lights) completely wrecked a drainage tunnel, burning four workmen. De Chambrier[25] gives a summary of accidents occurring in 1919.

On Aug. 11, 1919, a rail which was being lowered in No. 2 pit, jammed in the shaft. During the attempt to loosen it the cable broke and the rail fell to the bottom of the shaft, producing sparks which ignited gas in the oil sump which was then located at the shaft bottom. A slight explosion followed, injuring one man. One of four men caught in the bucket when it jammed midway to the surface, was killed by the smoke. The fire was brought under control by pouring large quantities of water down the shaft. Then 16 men were rescued who had taken refuge in a heading in which there was an air pipe.

On August 8, 1919, a fire, started by sparks made when a pyrite nodule was struck by a hand-pick, broke out in one of the galleries of No. 1 pit. The burning area was isolated by iron doors reinforced by 1.5 m. (5 ft.) of sand barrier. Nothing abnormal occurred for 5 days, when, on Aug. 16, following an inspection to see if ventilation could be established or the barriers moved nearer the threatened area, the "inevitable" explosion occurred. Two bore holes were then drilled into the affected area and 18,000 cu. m. (630,000 cu. ft.) of steam were forced into the area. On Sept. 12, when the fans were started to get an air sample, a second explosion occurred. After 12 additional days of "steaming" the fire was found to be extinguished. The Government inspection service then forbade reopening the mine until a second escape shaft had been sunk a year later.

From experience in the past, it is thought by the officials that in case of fire there are two alternative methods of procedure; either to maintain the ventilation, or to isolate the threatened quarter and erect sand barriers to lessen the effect of the explosion which they believe is almost certain to follow.[26]

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[25] *Op. cit.*, 90.

[26] A method that might be used to fight the fire other than using steam: viz., CO_2 injected through the compressed air lines, provided fusible plugs were placed in the air line between each face and the respective fire door.

Working Conditions and Wages

The miners worked in three shifts of 6 hr. each. These short hours were the same as at the Wietze oil mine, developed by the same company and by the Deutsche Erdöl Aktiengesellschaft, where the conditions for workmen were more difficult as regards temperature and disagreeableness. The fourth shift consisted of timbermen and repairmen. The working conditions at Péchelbronn were not at all bad, although the miners were covered with oil, and the oil-sand gas was sometimes objectionable. The mine was not hot and the air at the faces visited was good. The miners earnings averaged in August, 1923, about 20 fr. per day, equal in exchange at that time to $1.25, but in purchasing power in that district to about double that amount in the United States.

Production

The first of the present pits, No. 1, was sunk in 1916 and started operating in 1917. Within 3 months, the value of oil obtained by drainage into the development galleries had covered all the expenses of sinking. However, there was no competition and Germany then needed oil. During the year 1918, two other pits were brought in, which have been held in reserve. The sinking of two additional pits was being considered in August, 1923.

From the spring of 1917 to August, 1919, 36,362 m. tons of crude oil were extracted from the Clemenceau pit with its 3396 m. (11,200 ft.; or over 2 miles) of galleries. Each meter (3.3 ft.) of gallery length therefore averaged a yield of 10.7 m. tons (about 75 U. S. bbl.) of crude oil.

In 1923, the production from the mines alone was about 100 m. tons (700 bbl.) of crude oil per day. About 200 cu. m. (7000 cu. ft. or about 500 short tons) of sand were being hoisted per day, and piled on the dump awaiting treatment. About 2 or 3 tons of oil (14 to 21 bbl.) per day were recovered by drainage from the dump.

THE WIETZE OIL FIELD

The oil field of Wietze is situated about 45 km. northeast of the city of Hanover, Germany, and is controlled by Deutsche Erdöl Aktiengesellschaft ("Dea"), who successfully initiated the method of mining oil at Péchelbronn in 1916.

In 1917 the company was confronted with a decrease in production from the wells[27] to one-third the rate for the previous 8 years and, encour-

[27] In this field 1000 wells had been drilled within a few hundred meters of each other.

aged by the success at Péchelbronn, decided to attempt a similar exploitation at Wietze. Two shafts were put down and the systematic drainage of the sands by means of galleries was begun. Although the difficulties of mining, including the necessity of using the freezing process to sink through the alluvium were greater at Wietze, the company gradually worked up to a monthly production of about 900 m. tons (6300 U. S. bbl.) of oil in 1923.

Geology

The oil sands at Wietze were said by the mine officials to be principally of Cretaceous age, some being upper Triassic. The oil sand strata dip 6° or 7° northward. The outcrops are buried under a covering of alluvial material about 60 m. thick. Folding in the immediate oil fields is slight, but the beds appear to have been laterally compressed and faulted. The general system of faults runs N. W.—S. E. with a dip to the S. W. One profound fault to the south of the mine limited the workings. Sometimes when passing through a fault it is difficult to identify the bed beyond, as the beds have no distinctive characteristics, one from another.

It is believed that some of the oil came up along the faults and impregnated the sands and clays of the Tertiary diluvium. This would account for its presence here and there covering the water of ponds and swamps. The strata are said to be richest in oil near the faults. Very little gas was obtained from wells tapping the oil sands. Some salt water was encountered. The oil-bearing formations are underlain by earlier potash salt beds of Permian age and there are several potash mines in the neighborhood of Wietze.

In only a few wells of the oil field does the oil flow without pumping, but some of the wells had been pumping for 26 years.

Nature of the Oil

Locally they classify the oil as of two kinds, the "heavy oil," with a density of 0.939, and the "light oil" with a density of 0.886. The former was usually found nearer the surface, that is within 90 to 200 m. Both kinds of oil had a paraffin base, but the heavy oil contained only about 1 to 2 per cent. benzine. It was thought and hoped by the management that deeper oil sands, lying at about 300 to 350 m., would be found which would contain more benzine.

Production

The following table shows the production of oil in the Wietze field between 1908 and 1919.

Production of Oil in the North German District[28]

Year	Metric Tons	U. S. Barrels
1908	113,002	791,000
1909	113,518	794,600
1910	110,996	777,000
1911	98,639	690,500
1912	82,438	577,100
1913	71,174	498,500
1914	61,130	427,900
1915	55,919	391,300
1916	51,243	358,900
1917	43,616	305,200
1918	38,027	266,200
1919	27,353	191,500

The daily production from the mine in 1923 was about 30 tons of oil (210 bbl.), but it was still in the development state. This would be at the rate of about 10,800 tons or 75,600 bbl. per year. The management expected to increase the production to 4000 or 6000 tons per month, or 48,000 to 72,000 tons per year.

Character of the Sands and Clays

There were four beds of oil sand, each about 10 m. thick, interbedded with clay, occurring at the following levels: No. 1, 180–190 m.; No. 2, 200–210 m.; No. 3, 230–240 m.; and No. 4, 250–260 m.

The sands are not as coherent or solid as at Péchelbronn, but are more "mushy" in character. Moreover, the sands seemed to be more thoroughly saturated with oil and resembled an oil quicksand. The men at the face had to work standing on planks. It was possible at the faces visited to thrust a walking stick down into the floor for a foot or two. The sands were said to be clayey, but at the faces visited no clay was observed and the "grab" samples taken were free from it. The clay between the sand lenses is compact, but plastic (called "fat" by the Germans) and bluish-grey in color and is not shaly. It is naturally dry and quite hard, but the presence of any moisture, such as from humidity of the air, turns it into a more or less plastic mass. The clay had suffered a great deal of movement as indicated by small slips and even a hand specimen showed slickensided surfaces.

Character of Roof and Floor

The sand beds were usually so thick that the galleries were driven entirely in them, with both floor and roof of sand. This necessitates close and continuous lagging. At the faces and in the galleries visited, the timber sets showed signs of squeeze. Where galleries had a clay roof

[28] Dr. E. Kohl: *Gluckauf.* (Feb. 5, 1921).

it was apt to be very heavy and if the floor was clay, that was said to squeeze up.

Water in the Oil Sand Stratum

The absence of any great quantity of water under hydrostatic pressure is essential to the practicability of any method of mining oil sands. While there was a small amount of salt water at Wietze, there was nothing approaching an artesian circulation and it did not appear to be under high pressure although a depth of 270 m. (890 ft.) below the surface had been attained at the time of examination. The sands and clays are cut by veinlets of pure clay which run in all directions and which by their impermeability lock up the water in more or less local reservoirs. Hence any water pressures were local. The oil, as drained from the sands, contained about 6 to 7 per cent. of water.

Gas Pressure

Gas pressure had not been encountered during mining operations, and there was very little pressure anywhere in the oil field. Only a few wells had been flowing wells.

General Layout of Mine

As at Péchelbronn, the general arrangement of mine workings was that of a more or less rectangular system of drainage galleries. These consisted of the main drifts which were roughly parallel following the strike of the sand beds and which were connected by crosscuts at regular intervals.

Galleries and Gallery Linings

The main galleries were elliptical in cross-section and lined with brick. At the middle of the lining there was a course, sometimes two, of wooden blocks to take the squeeze. In the galleries visited the wooden blocks did not show much squeeze. With this type of gallery the bottom of the invert under the trackway served as the oil channel or had a drainage pipe in it. The other galleries were lined with wooden timbering of heavy construction. Cement guns had been used in places to gunite the timbers as a precaution against fire, but it was observed that a great deal of the gunite had scaled off.

Methods of Mining at the Face

As already stated the sands at the face were quite soft and oozy. The headings were advanced in the sands by a system of forepoling, with the face bulkheaded, a system which is much the same as that used in very soft ground in this country. (See Fig. 9.) Planks or sills had to be

put under the posts as well as across the face and only one plank was removed at a time from the face. In very soft sand headings, side spiling had to be driven in addition to the roof spiling. It was stated that much the same method is used in clay headings.

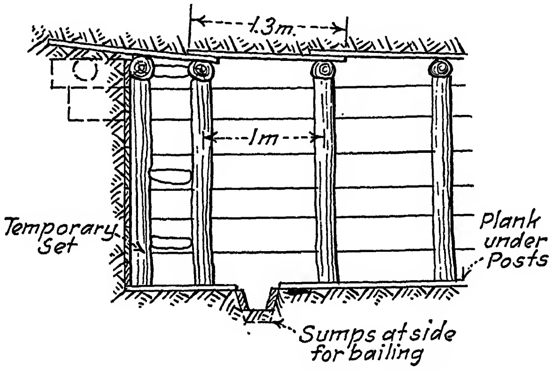

FIG. 9.—FOREPOLING AT WIETZE.

DRAINAGE OF THE OIL

It is believed by officials of the company that here, as at Péchelbronn, perhaps as much as 80 per cent. of the original oil content of the sands is unobtainable by drilling. Underground drainage galleries, according to von Hofer[29] get about half of the oil left in the sands after extraction by wells, because "pillars" of saturated oil sands, from which the oil will not drain, remain between galleries. In order to obtain complete extraction of all the oil, a systematic and complete removal of the sands themselves is required. This, however, was not practiced either at Wietze or Péchelbronn. He calls attention also to the parabolic shape of the surface of the oil in motion in the sands, when draining to a well or to a gallery.

CHANNELS AND TROUGHS

Each drainage gallery had either a gutter or a launder for the drainage of the oil to the sumps. Several launders suspended from the timbering were noticed, but their use was restricted to the shorter galleries where the oil was bailed into them from small sumps in the floor. (See Fig. 10.) They conveyed the oil to channels in the floors of the main galleries.

In some places long holes had been put in between two sand beds or drilled up to reach the top of a particular sand bed and drain the sands where these were thickest or have most oil. There was no regular position for these holes which merely served to accelerate the drainage from the sands which seem to be somewhat lenslike in character.

[29] Schachtteufen oder Bohren. *Pet. Zeit.* (Feb. 1, 1922), 113.

Sumps and Pumps

The main sump to which all the oil eventually finds its way was lined with brick. The oil is pumped through a bore hole to the surface. If there is any water, it is pumped with the oil, but only small quantities

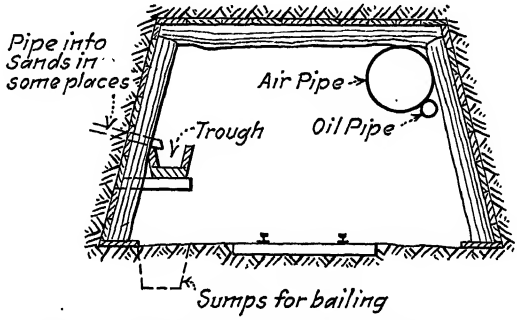

Fig. 10.—Section of gallery at Wietze.

of emulsion are produced. The pump room is circular and brick-lined. The pump is of the deep well type with the pump head on the surface.

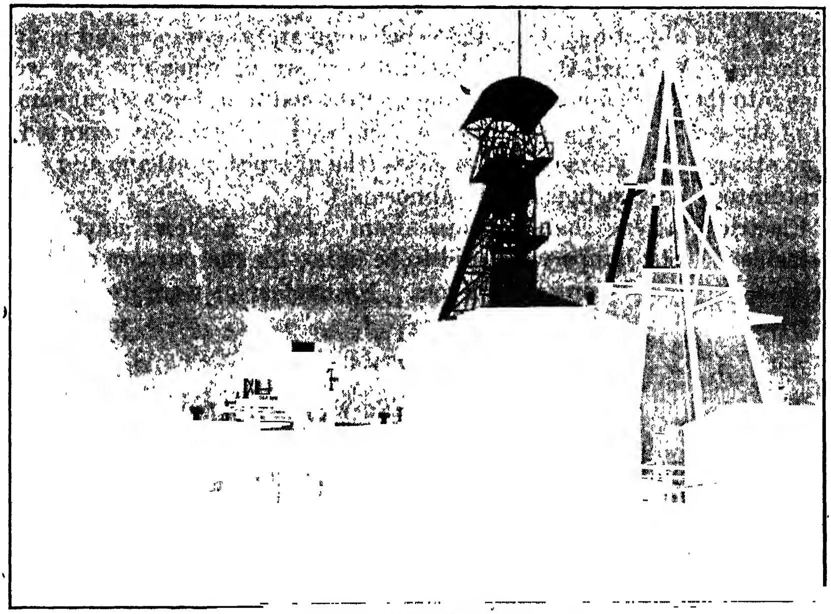

Fig. 11.—Headframe at Wietze shaft.

Transportation and Hoisting of the Sands

The sands were transported in tight cars holding 450 kg. (990 lb.) to the shaft, hoisted and dumped at the surface to await further treatment

in a washery of which the construction had been begun. There was a hoisting shaft and an escape shaft which were about 260 m. (866 ft.) deep. Cast-iron tubing is used as a lining for the upper 60 m. (200 ft.) through the water-bearing diluvium overlying the oil formations. The freezing system was used for sinking the shafts through this. The shafts are brick-lined below the tubing. The guides used were steel rails.

Head Frame and Top Works

The head frame of the hoisting shaft is of steel and has a substantial brick building housing the tipple, resembling the head frames and tipple buildings of German coal mines. The hoisting engine house and power house are substantial brick buildings, with fine machinery. (See Fig. 11.) The bathroom and attractive office building also are of brick and the general surface equipment is of high grade and somewhat larger and better than at Péchelbronn.

Mine Gases and Precautions

There was little or no gas pressure in the sands as mined and gas pockets in this oil field were said to be rare. The oil vapors here did not seem to be as objectionable as at Péchelbronn and the miners did not get headaches. Although the smell seemed very strong when one first went down into the mine, one soon became accustomed to it, but a slight smarting of the eyes from the oil vapor was noticed. There was reported to be no ethane nor hydrogen in the gases (the absence of ethane and other unsaturated hydrocarbons seems abnormal).

Electric battery hand lamps were used. A few single-gauze unshielded safety lamps were said to be in use for the purpose of testing the mine air for the presence of gas. No explosives were used nor are they needed.

Ventilation

The mine had a large centrifugal fan which was not working at the time of the visit. It was said that under atmospheric conditions then prevailing the natural ventilation resulting from the difference in temperature of the upcast and downcast was sufficient. The principal galleries had a ventilating pipe about 40 cm. in diameter and suspended from the caps, apparently for use only in emergencies. The air in the main return was said to average only 0.2 per cent. methane.

Fire Hazard and Precautions

There had been no fires. There was less danger from sparks at Wietze than at Péchelbronn because the sands were so soft that picks were not usually required. Wooden timbering was used at the faces and

in many of the galleries. Guniting of the timbers had been tried, but with none too satisfactory results because the gunite cracked off when the timbers took the squeeze. Mr. Gottfried Schneiders, a mining engineer of Berlin, advised,[30] against the use of wood in shafts. He also thought a long horizontal or inclined holes should be drilled in the sands to drain any gases or lighter oils, that drainage channels should be covered and that all the galleries should be equipped with fire and explosion-proof doors.

Number of Employes and Hours of Labor

There were 480 men underground in four shifts of 120 men each, and 120 men employed on the surface. The length of the shift was 6½ hr., "bank to bank" with 20 min. out for lunch. These hours became first established at Péchelbronn when it was considered that the work was hazardous and unhealthful. It was stated that the miners struck wh ena tentative lengthening of the shift was initiated. There were no government or legal restriction to the length of the shift, except when shaft sinking; in the latter case the shifts could be only 6 hours.

Working Conditions

The work was dirty, but the conditions were not as uncomfortable as in deep mines with high temperature and humidity. The miners worked practically stripped and were covered with the oil which was everywhere in the workings. They had complained to the officials that the oil is bad for the skin, but officials claimed that this was not so. The gases were not objectionable and the mine was not hot. The men preferred working in the mine to shaft-sinking which was too cold.

Bathing Arrangements

The surface equipment included a bathhouse of ample size and model construction. The arrangements for cleansing needed to be elaborate and provision was made to recover the oil with which the men were covered at the end of their day's work. When the miner came up he stripped and put his clothes in a hamper. Each man received ¼ l. of benzine to clean himself. He removed most of the oil with the benzine in one part of the room; the benzine and oil flowed through a drain in the floor. They were tapped off and were later heated to recover the oil; the miner then used soap in another part of the room before he went to the shower.

The miners' clothes were put into a rotary washer with benzine, and then were put in another washer with water and lye. Following this treatment they went to a drying room. No charge was made for this service.

[30] Gottfried Schneiders: Zechenbetrieb auf Erdöl. *Pet. Zeit.* (Aug. 16, 1923), 800.

Housing

The workmen's houses had two rooms, with kitchen, garden and stable. The rent for these houses at the time of the visit was 1,000 paper marks per month, a nominal figure. The company had built 100 double houses which lodge 200 families. There were also some larger houses built by the company in groups of 10 which lodged 80 families.

Wages

The men were being paid weekly at the rate of 2½ million paper marks per shift for miners, and 2.2 millions for surface men. The same wages were paid in the nearby potash mines. The wages increased about 50 per cent. per week. At the time of the visit butter cost 1.8 million marks per pound, and coffee was 4 millions per pound. At that time (latter part of August, 1923) the exchange on paper marks was lowering daily—their approximate value, per million, was from 20 to 30 c.

MINING COSTS

No data could be obtained, either at Péchelbronn or at Wietze, as to the cost of mining oil, nor could a definite statement of costs be found in any available publications, either French or German. However, a rough approximation based upon observations made in 1923, will be attempted in the following paragraphs:

The production at Péchelbronn was about 100 m. tons (700 U. S. bbls.) per day, which on a 300-day basis amounts to 30,000 tons or 210,000 bbl. per year. Also, one linear meter of gallery yielded approximately 11 metric tons of oil, or 23⅓ bbl. per foot. Hence a yearly production of 210,000 bbl. would require a total of 9000 ft. of heading advance, or 30 ft. per day. De Chambrier states[31] that the average daily advance per heading was 60 to 75 cm., hence a total daily advance of 30 ft. for the mine would require from 12 to 15 headings. The latter figure will be used in these estimates.

The cost of producing oil at Péchelbronn may be divided into the following items: labor, including underground and surface labor and mine superintendence; supplies; fixed charges, such as amortization and interest on investment; royalties and profits; and general expenses.

Labor Costs

Underground operations at Péchelbronn were divided into four shifts daily of 6 hr. each, of which three were employed in driving the headings, and one was used for timbering, repairs, etc. In 1923, the average wage for underground labor was 20 fr. With the rate of exchange then prevailing this was equivalent to approximately $1.25, although its purchasing power in the Péchelbronn district was about double this figure.

[31] Paul de Chambrier: Working Petroleum by Means of Shafts and Galleries. *Pet. Times* (1921), 210.

Using the figure of $1.25 as a basis, the following tables give estimates of the cost of labor employed in mining oil at Péchelbronn under the conditions prevailing in 1923:

ESTIMATED COST OF LABOR AT PÉCHELBRONN IN 1923

Underground Labor	Per Year of 300 Days	[a]Per Bbl. of Oil
Supervision: 2 foremen × 4 shifts @ $1.50	$ 3,600	
Advancing headings: 2 miners × 15 headings × 3 shifts @ $1.25	33,750	
Timbering headings: 2 men × 15 headings × 1 shift @ $1.25	11,250	
Renewing timbering: 2 men × 15 headings × 1 shift @ $1.25	11,250	
Building bulkheads: 2 crews × 5 men × 1 shift @ $1.25	3,750	
Haulage and track laying: 1 man × 15 headings × 4 shifts @ $1.25	22,500	
Maintenance of ventilation: 2 men × 4 shifts @ $1.25	3,000	
Maintenance of ditches, local pumping, etc.: 3 crews × 3 men × 4 shifts @ $1.25	13,500	
Hoisting (supplies, personnel, etc.): 1 shaft bottom man × 4 shifts @ $1.25	1,800	
Total underground labor	$104,400	0.49
Surface Labor	**Per Year of 300 Days**	**Per Bbl. of Oil**
Supervision: foremen	$ 1,800	
Hoisting: hoist engineers	1,800	
Hoisting: shaft top men	1,200	
Handling and dumping sand	1,800	
Ditching around dumps	600	
Blacksmithing and repairing	4,500	
Machinists and electricians	2,250	
Carpenters	2,250	
Roustabouts	2,400	
Power house: stationary engineers	1,800	
Power house: firemen	1,200	
Power house: laborers	2,200	
Total surface labor	$22,800	$0.11
Superintendence and Mine Office	**Per Year of 300 Days**	**Per Bbl. of Oil**
Superintendent	$ 1,800	
Clerk	300	
Timekeepers	1,200	
Total superintendence, etc	$ 3,300	0.02
Total estimated labor	130,500	0.62

[a] U. S. barrel.

Supplies

At an estimated cost of 15 c. per linear foot, a set of 8 in. round timbers in a gallery 6½ ft. high, 10 ft. wide at the bottom and 8 ft. wide at the top, would cost about $5.40. On the basis of two sets per meter, a daily advance of 9 m. would require 18 sets, costing $97.20 per day. Lagging 2 in. thick, on top, sides and floor, at $40 per 1000 board ft., would cost $86.40 per day additional. It is estimated that about one-fourth of the timbering was renewed annually.

Although some of the material in the bulkheads could be used again as they are moved forward to keep pace with the advancing face, provision must be made for some new material with each change. It is believed that $1 per yd. of gallery is ample for this purpose.

Ventilating pipe, compressed air pipe, and piping to carry oil from the local sumps, comprise another item in the cost of supplies for which the estimate is $2 per yd. of tunnel. Additional track and ties are also needed as the face advances, for which an allowance of $1.50 per yd. is made. Repair parts and renewals of machinery, equipment and tools are provided for at the rate of $20 per day.

The estimate for fuel is based on a plant of about 250 hp., using about 30 bbl. of fuel oil per day at $1.50 per bbl. to supply power for ventilation, pumping, and hoisting. Feed for 15 horses and the replacement of such as may become incapacitated is estimated at $15 per day, and an estimate of $20 per day for miscellaneous supplies completes the total.

Estimated Cost of Supplies at Péchelbronn in 1923

	Per Year of 300 Days	Per Bbl. of Oil
First timbers: (18 sets per day @ $5.40 per set)	$ 29,200	
First lagging: (tops, sides and floor @ $86.40 per day)	25,900	
Renewal timbers and lagging (25 per cent. per year)	13,800	
Bulkhead material: ($1.00 per yd. of tunnel)	3,000	
Pipe for ventilation, compressed air, local oil conveyance ($2 per yd. of tunnel)	6,000	
Track and ties ($1.50 per yd. of tunnel)	4,500	
Repair parts, and renewals ($20 per day)	6,000	
Fuel for power plant: ($45 per day)	13,500	
Horsefeed and replacement of horses: ($15 per day)	4,500	
Other supplies: ($20 per day)	6,000	
	$112,400	$0.53

Fixed Charges

It is assumed that the original investment is to be repaid at the rate of 10 per cent. per year. De Chambrier gives the following estimate for

the capital outlay for a mine with an annual production of 100 metric tons of oil obtained from a sand bed 2 to 2½ m. thick lying at a depth of 200 m. below the surface of the ground:

ESTIMATE OF INVESTMENT COST FOR AN OIL SAND MINE[32]

	Francs
Sinking two pits to 200 m. each, including lining at 4,000 fr. per m.	1,600,000
Steel for two pits	300,000
Two head frames	200,000
Rails, piping, tiling, cables, etc	600,000
Machine foundations and surface buildings	1,300,000
Two hoist engines	350,000
Fans, compressors, motors	350,000
Boiler equipment	400,000
Electrical plant	250,000
Tools, cars, etc	300,000
Miscellaneous	350,000
Total	6,000,000

This estimate was made in 1920, when the franc was actually worth 8 or 9 c. in American money, but allowing for differences in cost of labor and material, it would be advisable to rate the franc as equivalent to about 10 c.; making the total investment approximately $600,000, and the annual charge for amortization $60,000. Interest has been assumed at the rate of 6 per cent., but owing to the regularly decreasing principal (through amortization) the average annual charge will be 3.3 per cent. of the original investment.

ESTIMATED "FIXED CHARGES" AT PÉCHELBRONN IN 1923

	Per Year	Per Bbl. of Oil
Amortization @ 10 per cent. per year of $600,000 investment	$60,000	
Interest @ 6 per cent. (= 3.3 per cent. average on $600,000 investment)	19,800	
	$79,800	$0.38

ROYALTIES AND PROFITS

For completeness it has been deemed advisable to include in these estimates a charge for royalties and one for profits. The cost of royalties has been assumed at 12½ per cent., of the value of the crude oil at the mine, which in this case is taken to be the cost of production. Profits are estimated on the basis of 10 per cent. of the original investment.

[32] Paul de Chambrier; Exploitation du Pétrole par Puits and Galeries (1921), 97.

ESTIMATED "ROYALTIES" AND "PROFITS" AT PÉCHELBRONN IN 1923

	Per Year	Per Bbl. of Oil
Royalty (assumed at 12½ per cent. of $322,700)	$ 40,300	$0.19
Profits (assumed at 10 per cent. of $600,000)	60,000	0.29
	$100,300	$0.48

General Expenses

These include a proportion of the salaries of the general officers of the company, such as the president, general secretary, general manager, geologists, engineers, the selling and clerical force, and taxes and insurance.

ESTIMATED "GENERAL EXPENSES" AT PÉCHELBRONN IN 1923

	Per Year	Per Bbl. of Oil
Proportion of general management and general expenses (estimate)	$10,000	$0.05

RECAPITULATION OF ESTIMATED COST OF MINING OIL AT PÉCHELBRONN IN 1923

	Per Year of 300 Days	Per Bbl. of Oil
Labor	$130,500	$0.62
Supplies	112,400	.53
Fixed charges	79,800	.38
Royalties	40,300	.19
Profits	60,000	.29
General expenses	10,000	.05
	$433,000	$2.06

ESTIMATED COSTS AT WIETZE

The cost of mining oil at Wietze in August, 1923, is even more difficult to estimate than at Péchelbronn for two principal reasons; namely, the rapid descent from day to day in the value of the paper mark and the fact that underground development had not proceeded as far as at Péchelbronn.

At Wietze there were 480 men working underground, and 120 on the surface. These men received, according to the state of the exchange, from 75 c. to $1 per day; surface men got from 66 to 88 c. per day. The daily production of oil in August, 1923, was 35 metric tons.

Estimated Labor Cost at Wietze in 1923

	Per Day
480 men at $1.00	$480.00
120 men at 0.88	105.60
Total	$585.60
Estimated labor cost per metric ton oil	16.73
Estimated labor cost per bbl. oil	2.39

Although this is four times the estimated labor cost of Péchelbronn oil there are reasons which make it appear to be hardly a fair comparison. Wietze was a much newer underground development and had a smaller production of oil. There would appear to be no good reason why the ultimate drainage of oil per linear foot of tunnel should not be as great or greater than at Péchelbronn, unless the greater viscosity of the oil is a permanent handicap to drainage from a unit block of sands. Assuming this to be offset by the greater degree of saturation it seems probable that, after full development of the mine which will take several years, the cost per bbl. of oil would be as low as that of Péchelbronn.

Acknowledgments

The above paper is based upon data obtained by George S. Rice, chief mining engineer of the U. S. Bureau of Mines, during a visit to the oil mines of France and Germany in the summer and fall of 1923. These data have been supplemented by information gleaned from various French and German reports, especially those of Paul de Chambrier, General Director of the Mines de Péchelbronn; Messrs. Gignoux and Hoffman, the latter, geologist of the Péchelbronn Company; Herr von Hofer, of Vienna, and Herr Gottfried Schneiders, of Berlin, to whom the authors wish to acknowledge their indebtedness. Mr. Rice was accompanied on the trip to the oil mines by Lawrence Litchfield, Jr., who was of great assistance in taking notes, translating French and German reports and in compiling data.

Special acknowledgments are due the officials of the Mines de Péchelbronn and of the Deutsche Erdöl-Aktiengesellschaft, for the many courtesies and favors which made the visit to the mines so delightful and informative, particularly M. Weill, Ministare, Oúest Service des Mines de l'Etat at Strasbourg; Yves le Gorrec and Jacques de Chambrier, at Péchelbronn; E. Middendorf and Bela Szilasi, at Berlin, and Otto Klewitz, at Weitze.

Technologic Progress in the Oil Industry*

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(New York Meeting, February, 1926)

As an industry approaches stabilization, greater and greater stress must be laid on its technologic progress, which becomes a prime aid in improving its condition. The oil industry is tending toward this stage and hence its engineers must be pressed into greater service to prevent stabilization and stagnation from becoming synonymous, and to supply, either by improvements in processes or by new machinery, a means of obtaining savings in raw and finished products. Thus profits can be assured warranting continued investment.

It may be repeated, technology must apply a slight improvement here, a change there, or a complete renovation, to eliminate inefficiencies and make each dollar invested yield greater results. The profits of yesterday were made of rich strikes, of monopolized patents or processes, of control of prices and of markets. Sixty-five years of growth, of large investment, of healthful competition, and of expanding markets, have put the oil industry in the foremost rank of world industries. Just as the railroads passed a similar period of expansion and growth, and are being stabilized with reasonable profits, so the oil industry must take advantage of the efforts of its engineers and experts to promote its healthful growth and a better service to the people.

Herein is attempted a review of recent progress in which is presented the advances in exploration, production engineering, transportation, refining and greater efficiency in uses of petroleum, with a view not only to stressing important new processes and methods, but also the problems that press for solution. This review can only touch outstanding developments of recent years, but the several Petroleum Division symposiums of our program will present details worthy of attention.

Exploration

Geology offers directly little new in the technique of oil-finding—major stress being laid on improvement in subsurface correlation by means of foraminiferal studies, additions to the knowledge of which are being made, slowly but surely, and by means of mineral, and especially heavy mineral, determinations in well samples. The general application

* Petroleum Development and Technology in 1925, 19. (A. I. M. E. Pamphlet No. 1570-G, issued with Mining and Metallurgy, April, 1926.)

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of geophysics by means of torsion balances and seismographs, as a result of the experiments of the past two years, has proven the greatest new aid to geology. As if to prove the theory that man's ingenuity rises to the requirements of his age—just at a time when the geologist had reached the point where mapable geologic structures were becoming as scarce as dinosaur eggs, and far more valuable (despite our present apparent flood of oil production)—the physicist has offered the two types of instruments which will permit, under special conditions, the location of salt domes, faults and anticlines. There are three types of torsion balances, Eotvos, Bamberger and Oertling, but all are essentially the same as the original Eotvos in principle, varying only in portability, sensitiveness and the presence or absence of a self-recording photographic attachment.

These instruments give the differential in millionths of the density of adjacent portions of the earth's surface. Thus, beds of a higher density below one part of the surface (such as limestone-anhydrite-capped salt domes) may be distinguished from those of a lower density (such as ordinary unconsolidated sediments) surrounding them. The seismograph attempts to make use of an entirely different principle, that of pressure waves, refracted at varying velocities through different rock masses. Such pressure waves are actuated by the explosion of a small charge of dynamite (100 to 200 lb.) at or slightly below the surface, and the recording of such waves, usually photographically, by means of either one of two general types of seismograph—mechanical and galvanometric. The mechanical type has frictional difficulties, while the galvanometric type tends to magnify, to some extent, extraneous sound or pressure waves. With proper checks the latter type, due to greater magnification, offers the greatest possibilities, for it permits, by varying the amount of explosive, the separation and obliteration of most of the extraneous waves. Of this type, three machines, the McCullom, Karcher and Rieber seismographs, each of which possesses some special advantage, are now, or shortly will be, available for commercial use, and Eckhart is working on a fourth. These machines all require less dynamite than the mechanical, are equally portable and have recording cameras. The mechanical machines include the German type used by Mintrop and improved types used by the Humble Oil Refining Co. staff and Dr. Ricker.

Boring holes for the charges and using smaller quantities of dynamite eliminate the high cost of filling craters. The inclusion of proper time and air sound-wave devices, it is believed, will ultimately reduce the cost of and necessity for the elaborate land survey parties now in use.

Insufficient Experience with Geophysical Methods

A word of warning: While a combined use of these geophysical methods will determine the approximate outline and depth of certain structures, a sufficient experience is not yet available in event surveys

are negative by either instrument to condemn the territory in which no selective results are obtained. Results appear obtainable with the torsion balance which are impossible to check with the seismograph and until further studies are made and some of these localities drilled, the failure of such check by no means condemns. This must continue true until the effective maximum depths for each instrument and a large amount of research into variable factors, such as the reflective and refractive indices of various rocks, can be determined. Seismographs are practically valueless in most hard rock areas, where a repetition occurs of thin hard beds. However, since the bulk of the world's oil supply comes from younger beds, which often prove capable of study by this means, they must become important factors in oil exploration. The next step, and one being attempted by a number of competent investigators, must be the application of reflected sound waves by means of which it may be possible to determine depths of various strata, one below the other, especially such as are of appreciable thickness. The general principle of the sonic depth finder is here involved.

Following geology and geophysics, coring devices are becoming more and more important, and the new technique has largely to do with the improvement of double core barrels to be used in conjunction with rotary and even cable tool drilling, in order to eliminate doubt as to formations penetrated. Diamond coring devices have long been efficient for hard rock and these, with other means, are, except in special instances, adapted to shallow coring. While the diamond coring machine is being used somewhat for depth, special core barrels, such as the new modified or Elliott Alco barrel, in conjunction with a rotary, permits accurate coring results to depths of 7500 ft. and more. That accurate information as to the character of oil horizons, etc., is available by this means is most important when the great cost of deep holes is considered. The next important requirement here is the determination, by quantitative study, of the oil and water content of such cores; the Bureau of Mines will shortly publish some results that may prove helpful in this respect. A series of special papers on Coring, under the Production Engineering Symposium, bring out many facts on this subject.

The increasing use of diamond and other coring for shallow exploration will probably show a manifold growth in the next five years, first as a means of proving indefinite structure and second, as a check on geophysical exploration. Diamond coring will be more and more restricted to regions of rigid beds, and light portable rotary rigs, with the double core barrel, to wholly or partially unconsolidated beds and soft shales.

For details of subsurface correlation and geophysical methods, the symposiums by the American Association of Petroleum Geologists, to be held late in March, at Dallas, Texas, are recommended.

Recent studies by the U. S. Geological Survey of geothermic conditions showing increase of temperature in proximity to structure may prove an aid, as recently stressed by Dr. Thom. Water analyses are also proving a help though further research is necessary to make their use effective.

Production Engineering

This subject is young—very young—actually only some 30 years old, but has made great strides in the past few years and is already playing a heavy part in changing the economics of producing petroleum. It covers a broad range of subjects and reference can be made to only a few of the more important that are being used to reduce exploration hazards, to hasten completions and to increase the depth of wells, to increase production and to lower costs.

In the development of new pools more complete and precise records are being obtained, by coring and otherwise, so that complete records of water levels and other essential data are available, from the earliest stages of development, instead of being obtained as a post-mortem. Therefore, unnecessary drilling into water may be avoided and cementing back may be accomplished with precision, thereby saving many wells heretofore water-flooded.

The two outstanding developments in well-spacing of medium to deep wells in flush pools, where the oil contains much gas, have been:

1. A reversal of opinion regarding closely-spaced wells, as it has been demonstrated that, while first cost is greater, due to increased number of wells, their output is sufficiently increased to materially reduce the cost of lifting, and

2. The necessity of arranging wells staggered in alternate rows in proper relation to direction of rise of oil updip in order to partly avoid triangular blocks from which oil would fail to reach the wells.

This stresses the necessity for testing by coring the limits of production and for restricting the opening of new pools so outlined until the oil is needed. While, in a few instances, offsetting agreements can be obtained between competing companies, in the majority of cases cooperation remains difficult of attainment.

In addition to improvements in the shape of cable tool bits and heavier outfits and substitution of steel for wood drums, wheels, etc., a new type of bit has been developed by the Empire Companies' research department, which permits local circulations of fluid whereby more rapid drilling may be done through muddy formations.

Rotary Drilling

Real progress in the oil industry has come through the improvement of the rotary drilling rig and the perfection of rotary tools to answer

almost every purpose. Ultimately this process will replace most cable tool drilling. To this method must be attributed the rapid completion of wells and rise to peak production of recent major pools.

Improvements in rotary drilling have come through the production of heavier chain drive rotaries, with three-speed draw works, and the development of disk and roller rock bits. The rock bit readily penetrates hard rock and is more effective in deep drilling. Among new rotary devices for attaining greater depths the Hild differential drive permits control of drilling heretofore unattainable. It is perfected for use where electric power is available and is being adapted also for steam power. An aid of a similar purport is a steam-pressure strip recorder. A new type of rotary, making use of the drilling mud as a source of power, is the Russian turbinal rotary.

Improvement in steel derricks has been such that wood is gradually being replaced. Special studies have determined the best steel stocks for many types of oil equipment. Drilling bits have been improved by use of specially tempered steels or dressing with special drilling edges such as stellite.

Heavier casing and the use of a float valve for control in setting long strings are aids in deep drilling, as are a number of new fishing tools.

In finishing wells, high-pressure control heads and oil savers have proved invaluable in avoiding material losses of crude.

An electric water-witch, recently perfected by the Shell Co., ascertains definitely the points at which water enters a well. The device is based on the careful measurement of the differential conductivity of fluids.

New survey methods determine the extent of, and the direction in which, a well is out of plumb.

Cementing

Cementing of casing and cementing for the repair of wells to shut off water has received much attention. The Halliburton and Perkins processes are largely in use. Improvements are largely due to the use of portable cement wagons, arranged for the control of pressures up to 1400 lb., and the use of heavy compounds to weight the cement plug. Acceleration of setting is attained by several compounds. One of these—lumnite— is effective in 24 hr. Such compounds may prove worthless in case of heavy gas pressures.

The exact measurement of muds and weighting of same by heavy compounds admits of perfect control in drilling through high pressure gas.

Advances in Lifting Oil

In lifting oil, advances have been made through studies of the effect of vacuum, air, contained gas, extraneous gas, water-flooding and deep-well

pumping; and, while much has been accomplished, much remains to be learned. Through the application of the principles of the air lift to gas, the intermittent gas lift was developed by Lewis & Dunn for Mid-Continent conditions, and the continuous gas lift by McLaughlin & Jones (and a modified form by Templeton) for handling the large volumes of water-free oil of California flush pools. A compressed air drive is being used in some of the Mid-Continent and Eastern fields. Counterbalanced, long-stroke pumping has been developed to handle large volumes of liquid. Recent studies by Uren, of deep-well pumping problems, have been made available through this Institute the past year.

The effects of systematic shooting for increasing production are given you by H. B. Hill of the U. S. Bureau of Mines, while the results of water-flooding, so successful in the Bradford field, are presented by Professor Umpleby.

That continuous swabbing (as a means of lifting oil) has a harmful effect on the ultimate production of a well is now acknowledged.

For cleaning out wells, the compressed air method developed by Dunn has the advantage over the cable-tool method of saving time and reducing costs. The use of warm oil, to rejuvenate paraffined wells, has been successful in both California and Mid-Continent fields, in many cases practically doubling production.

Sanding of wells has been studied by John R. Suman and he has recently, in MINING AND METALLURGY, suggested some remedies. This problem affects the Gulf Coast and Venezuela wells and warrants much research.

Reference is directed to two papers offered in our Symposiums—"Deep Drilling Practice in California," by Robert H. Garrison, and "Improved Production Methods in California," by Frank O'Neill. These papers, together with discussions on gas lift, long stroke pump, etc., constitute a comprehensive statement of advances in production engineering.

Dehydration of oil-field emulsions has been advanced by three methods, (1) tretolite used with steam coils, (2) centrifuges and (3) electrical separation. Dow, of the Bureau of Mines, recently presented a summary of methods.

The increase of use of electric power in oil fields is notable, as important installations have been made in California, Wyoming, Kansas and Oklahoma. The only large direct installation made purposely for an oil field is that of the Midwest Refining Co. at Salt Creek.

For the most part, electric power companies are not yet aligned to supply power cheap enough to permit the change, and, moreover, it requires the installation of special motor-driven hoists, pumps, engines, etc. The great advantage lies in economy of power, as exemplified by

the use of portable engines, where heavier power requirements than normal are intermittently necessary. The improvement of spark-proof housings for motors, by both Westinghouse and General Electric, has been an important and necessary advance to permit their use at oil and gas wells where the fire hazard is great.' The Hild electric drive permitting accurate control of heavy loads, in connection with deep drilling, will bring this type of power more into use for such wells. Besides, as water powers become further developed and oil fields more nearly exhausted, greater use of electric power in oil fields must be expected.

Effect of Improved Methods

Improved methods are responsible for considerable increase in recent production and probably account for more than 25 per cent. of the production obtained from pools reaching their peak in the past three years and perhaps for one-fifth of the total production for 1925.

Major factors in increasing production are:

1. Closely-spaced wells, including inside locations, completed within a few months of the opening of the pool, thereby taking advantage of gas contained in the oil to obtain a maximum flow of oil from the wells.
2. Use of air and gas lifts.
3. Long stroke pumping.
4. Systematic shooting.

In general, the effect of these methods is to cause a larger immediate and cheaper production from new wells and earlier exhaustion of bonanza oil, with possibly greater ultimate production. Conditions involved in and extent of recovery of remaining oil are uncertain.

As to ultimate results: It is most important to distinguish pools in which water under great pressure surrounds the oil. Most of the large flush pools belong in this category. Hastened recovery under these conditions means earlier exhaustion of flowing and pumping oil and often in water-logging of portions of the sand.

Figures usually given of low recovery of original oil in the sands do not apply to pools of this type; for, instead of from 15 to 30 per cent. recovery, under ordinary conditions, these high-pressure pools probably yield 50 to 60 per cent., or even more, by methods now in use.

In any review of new processes, it must be emphasized that each is restricted to special conditions and, with additional use and research, these limits become more clearly defined. Thus, the gas lift system of the California flush fields is inadaptable to Mid-Continent pools with their waters and thinner sands, while the intermittent gas lift, in use in the latter fields, is inapplicable to such California production. Long stroke pumping, suitable to large mixed fluid heads, is useless where wells pump-off. Water-flooding, such as developed by Dorn and others, while

applicable under Bradford sand conditions, where the sand has been practically water-free and fresh water is available for flooding, can be used in very few districts. Sand-mining methods are likewise of restricted application.

There is necessity for careful study of each oil pool, or group of similar pools, so as to apply themost practical process. Many of these processes, while not patentable, are the result of long experimentation by certain individuals, whose services can usually be procured more economically than those of individuals attempting to duplicate their efforts.

Oil Sand Mining

Considerable work is being done on this important method of recovery of deposits to which usual recovery methods are no longer applicable. The Germans have developed methods which form a background for experimentation in this country and George S. Rice described their methods at the Mining Methods Symposium. Earlier in the year, Leo Ranney gave us a paper covering his mine-well process. A steam process has been developed by Georgeson for application to Athabasca tar sands. The important feature is not that any of these methods are able to compete at present with well production but that efforts are being made which will permit processes to be applied when well production in this country decreases sufficiently to warrant the next step which is oil sand mining.

Oil Shale

Next to be considered after oil sand mining is oil shale, which promises some 20 gal. per ton. In addition to laboratory experiments we are now to have two experimental plants, one in Colorado, under the auspices of the Bureau of Mines, and which will attempt a modification of the Scottish process, and a second, a 1000-ton plant to be erected for the N-T-U Co.

Transportation

Transportation in the oil industry embraces the use of pipe lines, tank cars and tank steamers and, collaterally, pumping equipment and storage. Of more than $9,000,000,000 invested in the oil industry, $850,000,000 are invested in transportation and, in addition, $175,000,000 in steel storage only partly owned by pipe line companies, and partly included in the previous figure. Recent advances in the technology of transportation have not been on a par with those in either production engineering or refining and yet, in many respects, a high efficiency has been reached.

Pipe lines are still largely constructed of soft steel pipe, usually 8 in. in diameter, and the chief advance has been in the mechanical methods used in construction; thus ditching machines and pipe-laying machines have materially reduced the force of labor required. Joints, for the most part, are still threaded, though heavier couplings are used. A few companies lay welded joints. The latter are primarily used for gasoline transportation and one of the largest lines for this purpose—48 miles—was laid by the Midwest Refining Co. a few years ago. F. Ray McGrew, of the Standard Pipe Line Co., Inc., will tell you about corrosion and protective coverings, as well as offer you a suggested method of new construction. He is of the opinion that only cement-encased lines are effectively protected. The use of "No-oxid," a patented chemical surfacing for steel pipe, gives results, but has not yet received sufficient trial.

The Goulds Manufacturing Co. recently conducted, in cooperation with oil companies, two research studies important to the pipe line industry. The first of these undertaken, in conjunction with the Union Oil Co. of California, was conducted by Professor Daugherty of the California Institute of Technology on the "Range of Application of Centrifugal Pumps," while the second, made in conjunction with representatives of several companies, is presented by Nelson B. Delavan in our Transportation Symposium and covers some causes of breakage in pipe lines. More of such work should be encouraged.

Where oil is of high viscosity and requires steam heating to transport, the old type of reciprocating pump still gives high efficiency. Vertical pumps of this type are used above 300 hp. The centrifugal pump is coming into favor for lighter oils.

Tank-car improvements have come chiefly in the form of heavier and explosion-proof cars for carrying gasoline and high-proof gasoline. An interesting development is the introduction of the use of trainloads of tank cars for long distance shipments of gas, as for example, that instituted by the Standard Oil Co. (Indiana), from Casper to the Gulf Coast.

Tankers for ocean shipments have changed primarily in rearrangement of upper deck and distribution of compartments for storing oil and refined products, and also in conversion, to a considerable extent, to oil-engine drive.

Efforts have been concentrated upon making steel storage tanks gas-tight, in order to reduce the evaporation of, and loss in, volatile products. In the case of stock tanks attention has been given improved bolted tanks which can be readily moved without cutting down the size as is the case with riveted tanks.

Refining

The Refining Symposium covers important phases of development in this fast-changing branch of the industry, so only a most general résumé

will be presented here. The two outstanding tendencies in recent refinery. practice are:

1. Making of cracking stills the central feature and all other processes secondary, due in part to the increased demand for motor spirit and in part to the rapidly changing aspects of cracking processes, whereby even low, gravity crudes may be partially and successfully converted.

2. The replacement of batch by continuous processes, and in this, the improved tube still as well as liquid phase cracking processes, play important parts.

A third factor is the extensive use of improved heat exchangers, which greatly reduce fuel bills and permit better heat control.

With 35 per cent. of our gasoline now produced by cracking, R. B. Day estimates a maximum efficiency for refineries will be reached when this is raised to 62 per cent. Gustav Egloff raises this to 75 per cent. This can be done only at large capital expense and a minimum of two years' time if all started immediately. It is entirely unlikely that this could be so rapidly effected for it involves restricting use of fuel oil.

The chief improvement in cracking has come through the development, first of vapor-phase and then of liquid-phase cracking in a continuous tubular still-type furnace in conjunction with a reaction chamber; this largely replacing the shell still-batch process. Another tendency is toward large units. Several processes of the former type are capable of handling heavy crude or topped residuum direct, although some of them require preheating in a tube still. In all of these, it requires a somewhat higher cost to crack the low-grade than high-grade crudes or light distillates. The batch still type cracks only light distillates successfully.

It is considered that 45 to 50 per cent. of cracking installations are already obsolete, due to research and keen competition.

The Tube Still and Its Uses

The efficient tube still has been evolved from the pipe still previously used for topping light fractions. The improvement has come partly in reducing the number of tubes, but chiefly in so building the furnace that the heat is equably distributed to all the tubes and can be kept under perfect control. The chief uses of this still are:

1. For continuous process distillation to replace shell still batteries.
2. As a preliminary heater for shell stills.
3. To handle crude or topped residuum preliminary to cracking.

One of its chief values is the low amount of carbon deposited during its operation. It can also be used as a cracking still.

Improved fractionating and bubble towers are another outstanding advance.

The Gray process is partially replacing sulfuric acid in treating oils and similarly the cheaper hypochlorite solution is replacing the established "doctor" solution for sweetening cracked gasoline. The Anglo-Persian refinery, in England, has adopted the hypochlorite solution. A minor change in practice at this same refinery is the use of bauxite to replace the fuller's earth so largely used in this country as a filter and decolorizer of oils and wax.

The use of centrifuges, both in elimination of sludge from lubricating oils and cylinder stocks and in certain improvements in wax separation, is important. The production of lubricants from asphalt base oils is one of the outstanding advances in the industry.

The chief changes in topping plants are substitution of tube stills for shell stills, use of baffled evaporators and new type fractionating towers. The erection of new small topping plants is becoming daily a more hazardous undertaking, for the large complete plants with cracking, lubricating and wax units hardly admit of such competition.

In addition to the extension of the use of electricity as power for refineries, some new possibilities lie in its use, for example: In the proposed catalytic process for treating cracked gasoline; in the subjecting thin low-viscosity oils to a glow discharge of high frequency electricity, to effect increased viscosity of lubricants, as used in Germany; and in distillation by means of a furnace in which a thin film of crude may be spread over a surface so as to be fractionated by an electric current. While none of these will compete commercially with processes now in use in this country, they do offer a field for further research worthy of encouragement.

Gasoline from Natural and Casing-head Gas

Production of such gasoline represents almost 10 per cent. of the total gasoline produced in this country, thus saving a product partly wasted heretofore. The chief improvement in its production has been in the use of cocoanut shell charcoal as an absorbent in the absorption process, which is rapidly replacing the compression process.

Standardization

Technologic progress is of two kinds—one purely mechanical, the other, one of method. To the former belongs the progress toward standardization of tools and equipment used in production, which is one of the most important results attained by the American Petroleum Institute and to J. Edgar Pew and Captain J. F. Lucey, who furthered such standardization, and the more than 400 co-workers much credit is due.

Results to date include specifications for steel and iron pipe for oil country tubular goods, on rig irons and cable drilling tool joints and on four standard sizes of locomotive type boilers, covering two classes of

working pressure. Practical agreement has been reached on standard rigs and derricks, and specifications will be completed in June. Tentative specifications have been adopted for wire rope and Manila cordage, and final specifications are hoped for within a year. Rotary drill pipe standards were adopted and two standards for rotary drilling equipment have been completed, as well as miscellaneous standards covering hoisting blocks, drilling hooks and standard taper for drive bushing. Progress has been made on pumping equipment, including sucker rods and engines, as applied to the production of oil. Tentative specifications, covering five kinds of belting, have been approved and it is hoped soon to have tentative plans and specifications for steel storage and production tanks. In addition, a new committee has been appointed to cover the matter of gaging and gaging practice. These results are available through the American Petroleum Institute bulletins and the specifications are being adopted at great initial expense by manufacturers, for it means ultimately a saving of millions of dollars to the industry.

Effective Use of Petroleum and Its Products

The supply of crude has more than kept pace with the record-breaking demand. Echoes of the conservation movement started some years ago in this country and kept alive by certain individuals and organizations, who have attempted to make estimates of the extent of our unrecovered and undiscovered resources, have reverberated until there has been awakened in the industry and in its chief ally, the motor industry, desire to determine both how refined products can be further perfected so as to give greater efficiency, and what can be done in the improvement of motor engines to make the use of these products more effective.

The perfecting of refined products is of course distinctly a refiner's problem. The perfecting of motor engines necessarily falls to the motor industry. Besides, the automobile owner is an important factor. However, cooperation is necessary to effect either of these improvements and, while research is active, certain economic factors must play a part in the solution of these problems.

With more than 20,000,000 motor cars and trucks in American use, with 367 refineries active the first of January, and with all motor plants now equipped with designs and dies for present type motors and cars, a huge capital investment would require replacement by the public, the refiner and the automobile maker. To effect this in a short period of time would be catastrophic, even in such a rich country as this. Besides, the petroleum industry must make better adjustment in production of its crude and storage of crude and products; also in restriction of the use of fuel oil to really essential uses. All of this requires time. Hence, the period required for further research to perfect petroleum products and

perfect petrol-using engines would also be necessary for financial readjustment and cannot be rushed.

Effective Use of Gasoline

In the Refining Symposium the papers on new gasolines and anti-knock compounds give some idea of the efforts of the refiners to produce more effective products, also of aids to increase gallonage per mile. Blending of straight run and cracked gasolines, production of more volatile gasolines for winter fuel, addition of certain compounds, such as benzol tetra-ethyl lead, etc., for increase of volatility and anti-detonation, are a few of the efforts to increase effectiveness.

Synthetic fuels being developed from coal in addition to benzol are methanol and that produced by the Bergius process, the latter not yet commercial.

The General Motors Corporation, as well as others, has been working on designs of high-compression motors, better carburetors, etc. For the effective use of crude, highly developed Diesel engines will fill certain requirements. The oil-electric locomotive, which can be operated at one-third the cost of steam locomotives, and which is just coming into use, is bound to become a large user of fuel oil. Considerable progress, too, has been made in developing crude oil burners for domestic heating.

Sources of Technologic Progress

Technologic advance has been especially fostered by a few institutions. Notable among these are the Federal Geological Survey, the Bureau of Mines, and more recently the Bureau of Standards, the California State Mining Bureau, the Illinois Geological Survey and the Pennsylvania Geological Survey. The Petroleum Division of the American Institute of Mining and Metallurgical Engineers and similar divisions of the American Society of Mining Engineers, the American Chemical Society, the American Society of Automotive Engineers and the American Society for Testing Materials contribute valuable technical papers. The American Petroleum Institute gives both technical papers and standardization specifications. The petroleum engineering departments of various universities, notably those of California and Pittsburgh, have undertaken research. Private research laboratories of the large refining companies, of motor companies, and those concerns developing special patents and processes have also contributed much toward these advances.

Publications of these organizations together with a few text books form the source material of petroleum technology in America. The *Bulletins* of the British Institution of Petroleum Technologists, as well as certain German and other foreign publications also furnish valuable material. Certain of the trade journals are to be commended for their publication of many worthwhile technologic papers. There are at present

two monthly indexes of current literature bearing on petroleum technology—that of the monthly MINING AND METALLURGY, published by the American Institute of Mining and Metallurgical Engineers, and the exhaustive index, published by the Bureau of Mines. Honorable mention should be made of E. T. Dumble, the father of production engineering in America; Dr. Joseph Holmes, founder of the Bureau of Mines, the first specifically technologic bureau of the Government; Dr. Bain, former Director of the Bureau of Mines; Sir Bovington Redwood and A. Beeby Thompson of London; Dr. William Burton, who first perfected a commercial cracking process; the Drs. Cross, for first making high pressure cracking effective; George F. Kettering, of motor research fame; Dr. George Otis Smith of the Geological Survey, and a host of others who have contributed to these advances. Credit is not only due to engineers who have reworked and perfected the tools and processes, but to that great group of practical field and refinery workers who have added something to a rule of thumb here, and deducted something there, laying the foundation for all advances made.[1]

Possibilities of Research

The field of research from which further improvements must come is very large and Dr. Van. H. Manning will outline some of its possibilities. In this connection more active efforts can be expected of the petroleum industry, through the American Petroleum Institute, as a result of the recent donations by John D. Rockefeller, and the Universal Oil Products Corporation.

It is desirable that better cooperation be established between the various practical, scientific and engineering associations, and an early joint conference of representatives from each is advisable for a more specific allotment of efforts and to avoid useless overlap.

Further interchange of ideas by members of each of these organizations cannot help but broaden them, as well as contribute toward a better correlation of the several branches of the industry.

During the past year the Petroleum Division of the American Institute of Mining and Metallurgical Engineers has made a start, in this direction, as follows:

1. By inviting members and non-members, including important executives, to its National Committee, and to participate in its sessions.

2. By holding with the American Petroleum Institute a joint group session on California production engineering.

3. By holding, at Tulsa, joint sessions of the Mid-Continent sections of the American Society of Mechanical Engineers and the American

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[1] It was intended to present a selected list of books and papers used in the preparation of this review, which lack of time prevented. Due acknowledgment is made to all whose work has been drawn upon for information.

Institute of Mining Engineers, each contributing alternate programs monthly.

What Can the Engineer Do for the Industry?

The engineer, as a thinker and planner, is becoming a well-established factor in the industry. What can he do to best further its work?

1. Cooperate with the practical man, who also has a definite contribution to make, and thereby facilitate the effectiveness of both.
2. Sprinkle technical knowledge with the salt of common sense so as to make both more effective.
3. Restrict his zeal so as to neither over-engineer nor over-machine the industry, avoiding expensive changes and installations unless fully warranted.
4. Participate in special and group research.
5. Contribute of his results by participating with papers and discussions, thus clarifying his own ideas and avoiding duplication of efforts by others, regardless of source.
6. Refuse to approve statements and reports unless warranted by sound reasoning and facts, thereby maintaining a standard of light and truth for the ultimate advancement of the industry.

Conclusion

A survey of the field of petroleum technology must impress all with the large amount of real advance made in the past 15 years, for in that period have come the problems of organized search for new oil pools, of deeper drilling, of curtailing physical wastes, of handling large volumes of crude and refined products, of improving production and refining methods and of the many other problems hereinbefore stressed. And it has witnessed the advancement and recognition of the scientist and engineer by the industry. All of this demonstrates that progress is being made by the industry toward conservation. There must be no contentment with results attained but there must continue an earnest endeavor for the solution of such problems, as the search for difficultly discoverable pools, better methods of recovery from sands, more efficient refining methods, more perfectly refined products, and finally, the more effective use of this unreplaceable bonanza resource.

Factors Affecting the Cracking of Petroleum*

By Charles L. Parmelee,† New York, N. Y.

(New York Meeting, February, 1926)

When Professor Silliman made his first examination of the newly-discovered Pennsylvania rock oil in 1859, he noted that different rates of heating produced different results in fractionation—longer heating produced larger percentages of light fractions. In 1862, it was found that when petroleum vapors are condensed in the still crown and dropped back into the hot liquid, the result of the distilling operation is materially modified. These early dates mark the beginning of what is called cracking petroleum and is explained as separating the component parts of the hydrocarbon molecules into more and smaller groups of the same elements. The phenomena were utilized for the next 30 years to increase the yield of burning oil. The procedure was that regularly practiced today in dry or destructive distillation.

Considerable scientific study was given to the subject from 1862 to 1905, and a number of patents were obtained, but the work done was not continuous or intensive until about 1905, when the increasing demand for gasoline attracted attention of petroleum refiners to the possibilities of cracking. The first equipment built and operated essentially for cracking petroleum on a large scale was placed in successful operation in 1912. The commercial practice is, therefore, less than 15 years old. Comprehensive intense study of the controlling factors during this period has resulted in rapid increase in our knowledge, and material improvement has been made in the practice of the art.

It is the purpose of this paper to outline briefly the state of our knowledge and to suggest some of the questions which are attracting attention of researchers at the present time.

The major purposes for which petroleum hydrocarbons, including natural gas, are or have been cracked are:

1. To obtain products used to increase the heat value and illuminant content of artificial gas.
2. During the World War, to increase the supply of toluol.
3. To increase the amount of products capable of conversion into industrial chemicals.
4. To produce pure carbon.

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* Petroleum Development and Technology in 1925, 329. (A. I. M. E. Pamphlet No. 1570-G, issued with Mining and Metallurgy, April, 1926.)

† Consulting engineer.

5. To reduce the viscosity of the oil, making it more available as fuel.
6. To modify the form of the wax content.
7. To increase the yield of motor spirit.

Gas Making

The use of petroleum alone and in connection with water gas for making artificial gas is of long standing. The controlling principles have been fairly well understood and the practice standardized for many years. This operation involves cracking the petroleum into hydrogen and other fixed gases, light volatiles which form illuminant and heat producing enrichers, tars and free carbon. If the cracking is incomplete, hydrocarbons in the motor fuel range may be produced. Developments of the past few years suggest possibilities of changes in viewpoint and objects sought, which may result in material changes in this art in the near future. Among these may be noted:

A. The preparation of special chemicals. During the war, toluol was obtained by stripping the gases in gas works, and special plants were installed and operated for the purpose of obtaining the maximum amount of this product. This practice has been discontinued, but the greater value of certain hydrocarbon molecules as bases for alcohols, acetates and other chemicals as compared with the value of such compounds as heat and light producers, suggests that stripping the gases at gas works may assume new importance. It is quite possible that for smaller plants the gas ultimately delivered may be a by-product instead of a major one.

B. Recognition of the value of cracked petroleum as an anti-knock motor fuel has already attracted the attention of the gas making industry, and consideration is now being given to the advantage of a combination of gas making and motor fuel preparation. Where the gas industry is combined with the distillation of coal, the availability of benzol and similar compounds add to the attractiveness of this line of effort. The combined making of water and oil gas, a process suggested some time ago, but not generally practiced, offers a possible large reduction in plant investment and makes it practicable to use heavy oils and residuums giving an increased amount of products available by stripping for petroleum chemicals and motor fuels, while at the same time maintaining the desired heat and candle power.

Chemical Manufacture

Two commercial plants have been installed at oil refineries to make alcohol out of materials obtained from pressure still gas. One commercial plant where the oil is cracked essentially for the making of the bases of alcohols, acetates, etc., is now going into operation. The value of portions of the gases from petroleum cracking operations as bases for derived chemicals, suggests the possibility that such operation may be quite

largely conducted for the major purpose of making unsaturated bodies convertible into organic chemicals. Ethylene derived from the gases of cracked petroleum is of considerable interest to the industrial chemist as the basis for the manufacture of derived chemicals. Investigations of ways and means of absorbing this gas are now being conducted on a large scale.

Apparently, most of the dyes, organic chemicals, etc., now produced from coal tar derivatives can be made from petroleum hydrocarbons. It seems quite probable that the oil industry will give this field considerable attention. The economic waste of burning hydrocarbons as fuel in competition with coal, when at a small cost they can be made into products worth many times as much, is apparent.

Carbon Black

Cracking natural gas is a chemical action resulting from incomplete oxidation and has for its major purpose the production of pure carbon or lamp black. It is practiced on a large scale. The same product can be obtained by cracking liquid hydrocarbon but as usually performed, such cracking results in the carbon atoms set free assuming less desirable forms. Legislation limiting the use of natural gas for making carbon black is reported to have created a shortage of this material. New methods of making it from petroleum are being discussed.

Cracking for Viscosity

Heavy highly viscous oils are rendered more fluid by heat cracking. At the same time some gas and naphtha are made but so far the amount has been small. That such heavy oils will yield anti-knock fuels and bases for chemicals is recognized. This suggests the probability that the cracking of heavy oils under controlled conditions may be practiced for a supply of motor fuels and chemical bases, and the desired viscosity of the resultant fuel oil will be secured as an incidental rather than a major purpose. This development may readily have considerable bearing on the oils available for fuel. The difficulty of using for fuel purposes an oil heavily loaded with free carbon as usually results from cracking operations may be met by better control of the factors affecting the formation of free carbon or by the use of improved systems of burning the residuum. The heat value of carbon from cracking is high, if its consumption as fuel can be secured. As ordinarily encountered or handled in the residuum, it acts like graphitic carbon and is an inert burden on the operation.

Cracking Wax Content

Cracking the wax content of oil containing amorphous wax is common practice in the refining industry. It results in increasing the gas, naphtha and coke yield and converting the wax into forms which can be more

readily removed by cooling and filtering. Modifications in refinery practice whereby the wax is removed in its original state without cracking, are in operation. The readiness with which the wax content may be cracked to produce gas and motor fuel makes it an open question whether the wax market or the motor fuel market offers the more desirable outlet for the hydrocarbons present as waxes. Price variations will cause changes in operation from time to time.

Cracking for Motor Spirit

By far the greatest present importance of petroleum cracking is in the production of motor fuel. In 1925, about 30 per cent. of the motor fuel production of the United States was obtained by cracking. By the end of 1926, the available cracking units should be able to supply about 300,000 bbl. per day of motor fuel from cracking, or about 45 per cent. of the total 1925 production.

Hydrocarbons available for this purpose are produced in operations directed mainly along other lines. Likewise, in any operation for the production of motor fuel there will be a certain amount of conversion into forms not available for this purpose. One of the major results sought in all systems of cracking petroleum for motor fuel, has been to hold the production of non-available by-products to the minimum possible. This condition may be changed in the near future by the development of the chemical possibilities of petroleum hydrocarbons.

The method of cracking desirable to be employed is affected or determined by many factors, but among the major ones are:

1. Character of charging stock.

2. Character of motor fuel desired, *i. e.*, whether a complete fuel or blending stock is sought, also the boiling point curve of available blending stocks. The anti-knock qualities of certain classes of cracked products is rapidly assuming major importance.

3. The outlet for the by-products often affects or determines the method or the extent of cracking desirable.

Cracking may be accomplished by heating alone, or by chemical reaction with and without heat. Except as the use of a catalyst may be said to be of a chemical nature, no method of cracking petroleum hydrocarbons has proven commercially practicable other than that of heating. The effects noted as due to the transmission of electrical energy seem to be largely, if not wholly, due to heat.

At the present time, there are 26 cracking systems in commercial operation, each regularly running 100 or more barrels of charging stock per day. There are also 28 commercial-size experimental or demonstration systems in occasional operation. It is difficult to devise a constructional or operative grouping into which these systems can be placed for general comparison. Such groupings are unsatisfactory, owing to the

fact that most systems are mainly overlapping variations in the combination of fundamental features. It will, therefore, be more helpful to consider these fundamentals and not to attempt to group the operating systems.

Cracking petroleum as commercially practiced involves three major steps:

1. Heating the molecules to the temperature necessary to cause rupture under the conditions provided for digestion. Overheating results in over-cracking with excess production of hydrogen and carbon.

2. Removal of the lighter parts of the cracked products and separation of them into fixed gases not condensable at atmospheric pressure and temperature, and cracked distillates.

3. Retreatment of the unvaporized residuum.

Use of Catalysts

The effect of heat on hydrocarbons and other compounds is materially modified by the presence of various elements and compounds having a so-called catalytic power. They may cause more active or complete dissociation of the components of the molecules; cause dissociation to take place at a lower temperature; affect the character of the products of dissociation; or assist in the regrouping or polymerization of the original products or the dissociated portions. Prior to 1905, a large part of the research was in connection with catalytic action.

While there have been many attempts to take advantage of the influence of catalysts in cracking petroleum, they have not been generally successful because of the costs involved. There are three commercial plants in operation in the United States using catalysts. One of these using anhydrous aluminum chloride is employed on a large scale for the production of saturated distillates and residuum. Two other plants crack the molecules while in the vapor phase with the object of obtaining the maximum amount of compounds available for anti-knock fuel.

With the increased interest in high compression fuels, the use of catalysts may become of increased importance. The principal determining factor appears to be the cost of operation. Perhaps the major advantages of the use of a catalyst is the ability to secure the cracking result at temperatures at or below the boiling point of the charging stock. Cracking by heat without a catalyst involves in most cases heating the molecules to above their normal boiling point at atmospheric pressure. Some of the very heavy hydrocarbons will break down if heated to temperatures below their normal boiling point. Because of the heavy carbon formation, such hydrocarbons are used only to a limited extent to increase the motor fuel yield, and are cracked mainly to reduce their viscosity and make them available for fuel oil, the motor fuel production being small in amount.

Use of Pressure

In 1925, about 98.5 per cent. of the cracked gasoline produced was the result of operations under pressure of over 20 lb. per sq. in. Of this, 73.5 per cent. may be classified as low pressure, 20 to 200 lb. per sq. in.; 15.9 per cent. may be classified as medium pressure, 200 to 400 lb. per sq. in.; and 9.2 per cent. may be called high pressure, over 400 lb. per sq. in. About 1.1 per cent. of the cracked gasoline was produced at pressures over 1500 lb. per square inch.

Pressure is employed for two purposes:

1. To raise the boiling point of the hydrocarbons so that they may be heated to their cracking temperature while still in the liquid phase.
2. To affect the character of the products of cracking.

If pressure is not employed, it is usual to heat the molecule in the vapor phase. This is relatively difficult and, for the purpose of motor fuel only, somewhat undesirable. Heat transference is materially decreased when the liquid is vaporized as the molecules are spread apart. Molecules receiving heat from the source are liable to be overheated or burned because of their inability to transfer their heat to other molecules. This condition is usually tempered by the introduction of steam which materially aids in transferring the heat and also aids in the velocity of flow of the vapors, thereby increasing the rate of heat transference from the source of heat. It has been held that the steam breaks down and the hydrogen combines to keep up the hydrogen ratio in the hydrocarbons and the oxygen combines with the freed carbon and reduces carbon deposition. In some systems, the steam is subjected to decomposing temperature before or after adding it to the oil vapor. From a practical standpoint, the major use of the steam seems to be as a heat conveyor. The use of natural gas, hydrogen and inert gas has been proposed to aid in the heating problem and also to effect chemical combination. No commercial use of such ideas is reported, however.

Among the reasons for the various pressures employed, may be included the following:

1. *Variations in Charging Stock.*—The lighter the stock (lower its boiling point) the higher the temperature required to produce rupture of the molecules in the same time. Therefore, the boiling point must be raised higher to permit heating in the liquid phase. To accomplish this, the pressure must be increased. There is considerable difference of opinion as to the most desirable character of charging stock. Some advocate closely cut stock of narrow boiling point range, generally a heavy kerosene or light gas oil. The other extreme is represented by those who advocate charging heavy crude directly to the cracking stills, taking off the natural and cracked products as an original blend. Local conditions as to available equipment, character of crude or stock available

at low prices, market for residuum, etc., have great influence in determining the procedure to be adopted. The increased yield of anti-knock molecules from the cracking of heavy petroleums is attracting attention. If the carbon deposition can be controlled or handled economically, this factor may affect the selection of the charging stock.

2. *Character of Products Sought.*—With certain limitations, the character of the products of cracking is affected by the pressure on the molecule at the time of rupture. The lower the pressure, the more complete the fracture and the greater the amount of permanent gas and free carbon formation. It is known that hydrogen will combine with carbon at very high temperatures and pressures. These are considerably above those used in any commercial plant in this country. One German plant works with such conditions. It seems probable that the combining activity of hydrogen is not a suddenly acquired facility when a certain combination of temperature and pressure is reached, but that there is an increasing tendency to that end as the pressures and temperatures are raised. It is claimed that considerable recombination of hydrogen does take place at fairly low pressures. The character of the major products and amount of gas and carbon formed is clearly affected by the pressures used. At very high pressure there is materially less free gas and carbon.

This recombining tendency is affected by the digestion conditions. Hydrogen in the nascent stage combines freely. Molecular hydrogen is inactive. This is the theory on which most developers are working in retaining all the products of cracking in intimate association for some period after their dissociation has taken place.

It is also claimed that the character of the product is affected by the pressure and temperature outside of the extent of cracking or recombination of hydrogen. Whether this is true or whether the results obtained are due primarily to the character of the charging stock, is not altogether certain.

The modern tendency is clearly toward the use of materially higher pressures and temperatures. Up to 1918 there were no commercial plants in operation using over 200 lb. per sq. in. pressure and the temperatures were generally under 700° F. Pressures of about 1800 lb. per sq. in. and temperatures of about 1000° F. are now in commercial use. Possibly some of this increased use of higher pressures and temperatures is due to the improvement of the engineering technique in construction, making the operation less hazardous.

Method of Heating

Except for some limited use of electrically-heated tube sections and internal electrodes which have been tried experimentally, all of the cracking heat has been obtained by contacting hot gases from the combustion of coal, oil or gas with the metal walls of the oil container, either tubes

or vessels. Heating the oil by bubbling it through another liquid, such as lead, has been tried, also submersion of metal tubes and tanks in molten metal but they have not proven commercially advantageous.

The selection of the fuel is largely dependent upon local conditions. A considerable amount of non-condensible gas is formed in most systems and this is commonly employed in heating the cracking zone either alone or in connection with other fuels.

Marked advance has been made in the design of combustion chambers and type of mechanism used in burning the fuel. Mechanical stokers are quite commonly used for coal. More effective use is being made of radiant heat.

Return of flue gases for tempering the hot gases is a modern practice. Furnace temperatures must not be too high or there is danger of over-cracking. Careful control of air admission and preheating of the air for combustion are becoming standard. Automatic regulation of fuel feed to oil temperature has promising possibilities of gain in efficiency. Very slight temperature variations have marked effect on the continued efficiency of the cracking operation. Where the control is close and even there is great gain in efficiency.

Heating in Tanks vs. Tubes

The first Burton stills were simply shells set over the firebox and batch-operated at about 95 lb. pressure. For about two years, all of the cracked motor fuel sold was so made. It was difficult to prevent over-cracking or burning because of the comparatively stagnant body of oil in the still and because of carbon deposition on the lower sheets. False bottoms were used to collect the carbon above the fire shell. Sweeps or stirrers were installed in the shells to agitate the oil and to sweep the carbon from the shell and keep it in suspension. Stills of this type are still in operation on a large scale.

The first commercial departure from the shell still was to follow boiler practice and suspend a bank of tubes in the combustion chamber below the shell. The oil circulated by heat convection through a drop leg, through the tubes and up another leg into the shell.

The next step was to separate the tube bank from the shell which was kept out of the fire zone or only sufficiently heated to maintain the body of oil therein at the desired temperature. Thermo-siphon circulation did not always produce sufficiently rapid flow of oil through the tubes to obtain the most efficient heat transference and to prevent carbon deposition, and forced circulation was developed. With the increased flow of oil and rate of heat transference it was found practicable to absorb sufficient heat into the oil to produce the desired cracking in a shorter time without overheating. Tube heating made it easier to control the amount of heat transferred and the maximum temperature of

the oil and, therefore, offered a means of limiting gas and carbon loss. It also permitted the safe use of higher pressures and temperatures.

Heat cracking is a function of both time and temperature. It may be produced by repeated heating to a given temperature, by rapid heating to a higher temperature and by prolonged heating at a lower temperature. With the development of the tube bank, therefore, came variations in the length and size of tube used, rate of flow in the tubes, temperature of the heating flue-gases, once through and recycle operation, and the use of digestion or soaking chambers where the oil is maintained at the cracking temperature for periods varying with the different systems.

The present tendency is strongly toward the exclusive use of tubes for heating the oil with digestion for varying periods. In 1925 over 70 per cent. of the throughput was so heated. By the end of 1926, it will probably be over 80 per cent.

The size of the tube has been affected by several factors. Small tubes require higher velocities for the same throughput, but this results in increased rate of heat transference and reduced carbon deposition. The need of higher pressures has also tended to keep down the size of the tube.

Ease of cleaning and necessity for beam strength has tended to make for larger sizes. The present usual range of sizes of 2 to 4-in. internal diameter represents personal equations rather than scientific determinations. There is some use of smaller tubes at very high pressures.

Increased rate of heat absorption by the tubes has been obtained by the use of ribs and collars of metal which increase the available heat absorbing surface per foot of tube. This is made possible because of the greater rate of metal to liquid heat transference over that of gas to metal. Armored tubes are also used in vapor phase work although the advantage is not so well marked. It is also claimed that the armored tube has greater strength.

Digestion

The time necessary to effect the desired degree of cracking is a factor of the rate of heat transference from the source into the cracking stock, the maximum temperature of the stock, and to a less extent of the pressure. The modern tendency is to heat the oil slowly and to keep it in the heating tubes only long enough to absorb the necessary heat and to permit the reaction or cracking to take place mainly in large vessels variously known as bulk supply drums, vaporizers, expansion chambers, etc. In most cases it is desired that all or the major part of the vaporization and separation of carbon take place in these chambers which are maintained at full pressure. In some systems, the pressure is reduced between the tubes and the vaporizer and, in others, an intermediate liquid soaking drum kept at full pressure is used between the tubes and the vaporizer.

Vaporization

In most of the systems that are now in use, vaporization of the cracked hydrocarbons largely takes place in a tank containing a bulk of liquid and in which the carbon set free is deposited. Some of the vaporization takes place in the heating tubes, but it is generally the desire to hold this to a minimum in the liquid phase operations.

Some systems carry the entire charging stock through in one mass of mixed liquid and vapor and cool it all to atmospheric temperature and pressure, later reheating the cracked stock or synthetic crude at atmospheric pressure for vaporizing the light portions. This practice is not as common now as it was a few years ago, but it is still largely followed. Its advocates claim a better character of the resultant products as a result of more complete digestion.

As in simple distillation, the vapors evolved in cracking operations mechanically entrain and carry over considerable portions of products heavier than are available for motor fuel. From the early operations in 1912, it has been the custom in most systems to cool the vapors and reflux some of the heavy ends back before condensing the overhead distillate. There was not complete separation, however, and for a number of years the pressure distillate was all much too heavy for motor fuel and had to be rerun at atmospheric pressure, yielding from 20 to 60 per cent. available naphtha. The bottoms were usually used as fuel oil. About 1916, it began to be the practice to add the pressure distillate bottoms to the pressure still charging stock. There has been some increase in this custom, but in general it has not been found advantageous. With the use of higher pressures and temperatures permitting the ultimate cracking of this cut, more recycling of this fraction may be practiced.

The modern tendency is to reflux the vapors more completely while under pressure. It is a growing practice to use bubble towers and secure a large part of the overhead as finished stock in the original operation. In some systems, the reflux is mixed with fresh stock fed to the tubes for heating and cracking; in others, it is cooled and then added to the store of fresh charging stock; in others, it is added to the bulk liquid supply undergoing heat treatment. The controlling factors affecting this part of the process are the working pressure and temperature. When very low pressure is used, this light stock cannot be kept in the liquid phase and must be removed for effective operation. When low temperatures are used, this reflux stock largely circulates through the apparatus, its major value being as a solvent to keep the carbon in solution, though it is probable that some of it is cracked as a result of repeated heating.

Treatment of Residuum

The first cracking systems were batch operation, the entire body of oil being kept under heat and pressure until the still bottoms had been

reduced to what was considered the optimum degree. At an early date, it became common to add fresh stock during the run to make up for the stock removed overhead. This added stock was used as a cooling medium for refluxing the vapors. It was also common to bleed or draw off such portion of the heavier liquids in the system as the construction or operating conditions permitted.

For several years, all of the residuum in the still was removed at the end of the run. At first, the liquid was used for fuel oil. Later it was run down to coke at atmospheric pressure and the gas oil was returned to the cracking system as recycle charging stock usually mixed with fresh material. About 10 years ago, methods were devised for filtering or straining the residuum while under temperature and pressure, thereby removing the tars and suspended carbon, the filtered stock being returned to the heating zone. This practice has been improved and is followed in certain systems. Other systems continue to draw off the heavy liquid and remove it from the cracking system, returning only the refluxed heavy portion of the vapors to the cracking zone.

When catalysts are used, the residuum is frequently available as lubricant stock and it is claimed that ordinary cracked residuum can be so used. It is usually expensive to treat such residuum for color, however, and it is the general practice in non-catalytic processes to use the residuum as fuel oil or run it down to coke.

Production of Hydrogen and Coke

Every cracking operation tends to produce some free hydrogen and some carbon. The relative production of these by-products has been used as one of the bases of comparing the operations of different systems. The setting free of either element may disturb the hydrogen-carbon ratio of the remaining compound and affect either its value as a standard or anti-knock fuel or its tendency to go off-color and to form gums.

Hydrogen set free is customarily used as fuel gas. There does not seem to be any present use offering greater value.

The carbon largely settles out of the oil as coke. Various uses of coke from cracking have been proposed. It is usually burned for fuel, and is easier to consume than the carbon content of the residuum. It has not been generally found desirable for electrode construction. Briquetting the coke for domestic fuel is proposed. One system proposes to briquet under very high pressure without the use of any added binder. The use of pressure still coke as powdered fuel is now being tried in commercial practice.

Treatment of Pressure Distillate

Where ordinary pressure distillate containing considerable proportion of heavy ends is made, it is rerun at atmospheric pressure, and the

naphtha is ordinarily blended with natural gasoline. The blend is subjected to the adopted method of treating, steam stilling, etc. Refinery practice varies considerably as to the order and method followed. Discussion of this phase of refinery practice is to be presented by others, but the influence of the cracked products on color and gum forming constituents is to be noted. What the causes and remedies of these conditions may be, is still somewhat in controversy.

The use of fuller's earth in the treatment of cracked distillates is attracting considerable attention. One method of contacting the vapors with earth is in commercial use and another is proposed. With the introduction of the bubble tower and the recovery of end point distillates in the original operation, it has been found practicable to treat the distillate by earth contact while in its original vapor form and so avoid or reduce the treatment of the distillate after condensation. This method seems to have great possibilities.

Present Problems

The cracking art today is very highly developed as compared with its state when Dr. Burton demonstrated its commercial practicability. There are many factors, however, which are known to exist, the cause and effect of which are not clearly determined. The next few years will probably see considerably better definition of the fundamental principles, and as a result, many changes are to be expected. Among the points to which attention is now being directed are:

1. *Most Efficient Character of Charging Stock.*—This point has been mentioned. It is quite probable that local conditions are so widely at variance and so subject to changes as to prevent any generally uniform practice, but there is no question that more careful consideration of this point would be helpful in many cases now in operation. The demand for anti-knock fuels affects this question and may readily control decisions in connection with the stock and methods of treatment.

2. *Method of Heating.*—The use of powdered fuel offers an attractive means of using low-grade coals and still carbon. In most cases, there is still considerable room for improvement in design of combustion chambers, control of furnace temperatures, utilization of radiant heat and exhaustion of available heat in flue gases. The advantage of electricity as a definitely controllable heating means is recognized, but its cost has been prohibitive. With the desire for certain products best produced at very closely controlled temperatures, it is possible that heating by hot gases to just below the cracking temperature and then passing the heated oil through an electrically-heated cracking zone may be practiced. A system employing these essential features has been tried for motor fuel, but it was too expensive to operate for that purpose.

3. *Oil Containers.*—Tubes seem to have greater advantage for heating containers than tanks. The optimum size is open to discussion. The use of tube cores to reduce the stream flow to a thin shell of liquid has been proposed and in one case tried out in a commercial plant which was not successful for other reasons. Cores are used in one vapor-phase system.

The practicability of increasing the heat-absorbing surface by means other than by the armored tubes now in use are being discussed and will probably be given consideration.

Special alloys and surface treated tubes have their advocates, in some cases because of the increased resistance to corrosion, and in others, because of the reduced tendency of deposited tar and carbon to adhere to the walls.

In the construction of the larger vessels used for digesters, vaporizers, towers, etc., there is considerable difference of opinion as to the relative advantages of riveted and welded joints, and between the advocates of hammer weld and electrical weld. For very high pressure, vessels forged like gun barrels out of solid billets are used. In some systems, riveted joints with welded edges are employed. The effect of welding temperature on the resistance of the metal to corrosion is not altogether clear.

Cracked vapors are highly active in the corrosion of the steel used for containers. Instances of loss of $\frac{1}{16}$-in. metal per month are not uncommon. Special steels and alloys have been given considerable attention in the effort to reduce this action. Safety demands greater attention to the corrosion of vapor chambers and pipes than has always been given.

As the cracking temperature is ordinarily at the point of rapid reduction in the elastic limit-curve of steel and a modern practice is tending to higher pressures, there is a large field for study of ways and means for lengthening this curve. Some steel has been imported which purports to hold its low temperature strength to over 1000° F. Steel makers in this country will undoubtedly find an important field for study in this connection.

Clearer knowledge of the causes of corrosion and better defined information as to the effect of chlorine, sulfur, etc., and the economic value of their avoidance or removal from the charging stock are desirable. This also concerns the problem of treating the distillate. If sulfur is removed from the charging stock, not only will the corrosion problem be affected but the difficulties attendant on the combination of sulfur with the unsaturated hydrocarbons will be avoided.

4. *Length of Digestion.*—The advantage of long maintenance of oil at low cracking temperatures is urged by many. Periods of days are suggested instead of minutes. The disadvantages are the costs and danger of large amounts of heated materials getting on fire.

5. *Treatment of Cracked Products.*—In connection with end point refluxing, treatment of vapors as originally formed, recracking of refluxed

vapors, and treatment of residuum, etc., enough has already been said to suggest the trend. Unfortunately, there has been little scientific study of comparative results except by advocates of some system. There is a great need for definitely conclusive and generally usable comparative data.

Patent Situation

In this art as in any other, there has been extended effort on the part of researchers and developers to protect their ideas by patents. Over 2500 United States patents have been issued and about as many more applications are pending, which more or less purport to affect the cracking art. Not a single cracking patent has been adjudicated as yet, and while there is considerable litigation pending none of it promises definitely to clear the much confused situation.

Not one of the licensing concerns has a clear undisputed field and, on the other hand, it is held by some that no one has any legal rights of a broadly controlling nature. The Attorney General of the United States has questioned the scope and validity of all the major patents of the so-called Patent Club, which includes the Burton, Cross, Ellis, Holmes-Manley, Adams and other patents. It is not probable, however, that in all of these patents the courts will fail to find any merit. On the contrary, it is to be expected that some of them will be held to have sufficient breadth to create controlling rights to certain forms of practice.

Conclusion

In the use of the petroleum cracking art, as in any other, success demands a thorough knowledge of the existing local conditions and an intelligent application thereto of the best information available, always subject to the tempering of practicability. There is no mystery about its application, but there is great misunderstanding among its users. We know how to utilize some of the fundamental principles, but we are far from knowing the best method and the next few years will probably see great advances in efficiency and economy. Standard practice today may be obsolete tomorrow because of new developments in methods available and objects to be obtained. There is room for better engineering in this as in most other arts. It must be exercised, however, with due regard to the patent rights of others.

Basic Changes in Refining Processes*

By Walter Miller,† Ponca City, Okla.

(New York Meeting, February, 1926)

The remarkable strides which have been made in recent years in the art of petroleum refining are not fully apparent until we attempt to compare the scientific status of the industry today with the basic conditions existing a decade or so ago. The spotlight has played so strongly on the development of cracking that many of the wonderful steps brought about in other departments have crept upon us unawares.

A number of causes have contributed to this highly accelerated rate of progress. Constant effort by both companies and individuals to obtain the honors and prizes rewarding new discoveries has played its usual important part in our industry as in others. Competition has applied its continuous pressure on the refiner to reduce costs and losses and increase the proportionate yields of the higher-valued fractions. Advances in the engineering professions and growth in chemistry and allied sciences have contributed much. But there are other factors which deserve particular mention. The rapid development of the internal combustion engine, with the rigorous quality requirements and large quantity consumption of both fuel and lubricants has been a tremendous influence in adding impetus to natural progress. The teaching of petroleum principles and practices in many of our best universities and technical schools, and the calling into the industry of ever greater numbers of scientifically trained men to supplement the efforts of the practically trained operators, as well as to apply at close hand the fundamentals of science as a life work, has had and is having an incalculable influence.

Important Improvements

Let us consider briefly some of the most important improvements: First, distillation, which constitutes the greatest single phase of refinery operations, even with cracking excluded; that phase of distillation being handled as an individual subject in this symposium. Today we are distilling crude oil and redistilling by-products at a rate very much faster than was possible with the lesser knowledge of the art which we possessed

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years ago, at rates which in the most modern installations are two or three times that of former capacities.

Improvements in firebox conditions, making not only for greater efficiency but for considerably higher heat transfer rates through the firesheets are a factor in this increased capacity, with more progress still to be made.

Radiant Heat and Heat Exchange

One of the newest developments along this line involving much broader application of the principle of radiant heat, is known as the Cannon furnace. Square-shaped tunnels between a foot and a foot and a half in width and height and made of carborundum slabs, are placed in the combustion chamber with a separate burner for each. The combustion takes place mainly within the tunnels, heating their walls to radiating temperatures. This type of furnace is being used now in a number of plants, some places experimentally and in others in a comparatively large way.

The principle of heat exchange is being applied on a greater scale than ever before. While up to a few years ago heat recovery had been practiced in many European and foreign refineries to a greater extent than in the United States, higher fuel prices here and the development by engineering specialists of efficient and economic equipment is rapidly bringing about a greater use and realization of the value of this principle.

Internal Flues for Cylindrical Stills

A comparatively new practice in this country is the application of internal flues to cylindrical stills: A considerable number of tubes from 4 to 6 in. in diameter in the lower half of the still, so spaced as to have them all below the normal oil level, allow the combustion gases to make another pass against the heating surface, on the well known principle of the return tubular boiler, adding to both efficiency and capacity. This principle has been employed by many foreign refineries where high fuel costs obtained, the only difference being that one or two large tubes instead of 20 or 30 small ones, were generally used.

The various improvements in distillation methods on cylindrical still batteries have been stimulated by the competition of the pipe still, which has some advantages over old distillation methods and equipment. It is in connection with pipe still installations that the principles of vapor and liquid heat exchange have been developed to a high point of perfection. The radiant heat value of the gases of combustion have also been utilized to a larger degree on this type of installation. Many plants are being installed and it is now a well established factor.

Vacuum Distillation

Vacuum distillation, which has had a limited application since the early beginnings of the industry and lay dormant until about 1920, has undergone a new development. While the old method was to operate under a vacuum of from 20 to 27 in. of mercury, a patented process now being used quite extensively in the manufacture of lubricants from asphaltic and naphthenic base crude oils, is operated so near the zero of pressure that the gages show normal operating pressures of from 3 to 7 mm. absolute. In addition to whatever refining advantages it possesses, the mere building of commercial equipment in which such high vacuums could be maintained constitutes a remarkable engineering achievement.

The reduced pressure enables the distillations to be carried on at temperatures well below the danger point of cracking. There are quite a few commercial installations, all of them, I believe, on asphaltic and naphthenic base crude operations, on which type of material the process has had its main success so far. A much larger yield of the valuable lubricating oil fractions is obtained, and in addition the products themselves are of higher quality, and require much less chemical treating and decolorization to finish.

It is difficult to say at this time to what extent this method will be applicable commercially to the making of lubricants from paraffin and semi-paraffin base crude oils, but its usefulness in the case of the other type of crudes mentioned seems quite well established.

Fractional Condensation

The most important single factor in bringing about the present high standard in distillation practice has been fractional condensation. Improvements in fractionating methods are secondary in importance only to the cracking plant in increasing the yield of gasoline from crude oil. The story has been told many times of the stillman in Newark, N. J., refinery in the 1850's, banking his fires preparatory to spending an hour at home for luncheon, coming back several hours late and finding a distillate issuing from the tail stream very much lighter in gravity and color than he expected or than the stream was when he left. That incident, which is supposed to mark the starting point of cracking, was also a milestone in fractional condensation. It is a far cry from that day, of depending on the uninsulated top sheets of the still, to the elaborate system of heat exchangers, partial condensers, and reflux and end point products return methods used in connection with the modern complex tower systems. And what a difference in results! Elimination of the necessity of rerunning, ability to operate stills at capacities limited only by possible heat transfer rates and resistance of materials, and a close separation materially raising both the quality and yield of the desired product or products.

The influence of the requirements of the internal combustion engine on this question has been most marked. While my personal experience does not go back to the time when gasoline was burned under boilers and stills and was surreptitiously allowed to leak away, as some of the stories have it, I can well remember the practice of excessive air agitation of kerosenes for the purpose of raising the flash point to the legalized minimum, rather than depending on the stills to do this.

We have seen developed and tried countless schemes to take advantage of the fractional condensation principle. Enlarged domes or still heads, multiple and lengthened vapor lines, air-cooled tubular towers, cylindrical towers filled with "crocks," rocks, brickbats, scrap iron, and plates, perforated and unperforated, of every conceivable kind. Today the refiner's problem is to keep the flash and fire points of his kerosene products down to the maximum permitted by the buyers.

The Bubble Tower

One of the latest fashions in fractionation is the bubble tower. While employed for generations past in the alcohol, benzol and allied chemical arts, it did not find application in the petroleum industry until within the last few years. Its superiority over many of the forms of towers already in use was so marked, however, that it was quickly established on a very firm footing and is already in very wide use.

A still newer thought in the process of development is the application of centrifugal force to the vapors. One line of investigation being pursued is that of a low centrifugal speed with a full complement of special plates, another the development of a comparatively high peripheral vapor velocity in a small chamber without the use of plates.

Decolorization and Purification Methods and Equipment

Another outstanding example of the progress in the industry during the last few years is the improvement effected in the methods and equipment used in decolorization and purification of petroleum products. It is not that new chemicals are being used. Nothing has yet been found to replace satisfactorily that old standby of the refiner, sulfuric acid and aluminum silicate, to give fuller's earth its more scientific name, and today it is of greater proportionate importance than when it replaced the use of animal charcoal 20 or more years ago.

Closed System of Chemically Treating Gasoline

The substitution of the closed system of chemically treating gasoline for the old method of acid spray and air agitation in comparatively open top containers, permitted a more intimate contact between chemical and hydrocarbon, closer control of the operation with less chemical

expense, and reduced evaporation losses to practically zero. The lessened use of chemical in turn reduced the losses both in the form of sludge and polymerization, the latter being a very considerable item, especially in the case of cracked products.

Vapor Phase Purification

Vapor phase purification is one of the newest commercial developments in the treatment of gasolines of the cracked type. It must be remembered in this connection that the industry has been seeking for a score of years to develop a practical and economical chemical purifying process which could be applied to the gasoline fraction without the necessity of subsequent redistillation. So far we have been unsuccessful. We still find that the unsaturated gasoline fractions when brought in contact with sulfuric acid, besides suffering excessive sludge and polymerization losses, form sulfo compounds unstable in the presence of heat and light. These will break down in the motor with accompanying gum formation and corrosion, if the refiner does not eliminate the possibility by a heat treatment, usually in the form of a redistillation. This question is of increasing importance today because of the development of some cracking processes to the point of making gasoline boiling points direct from the unit. Much of the advantage of making gasoline direct from the cracking still is lost if the product must again be distilled.

It is the solving of this problem which constitutes the greatest claim of the inventor of the vapor phase method of treating cracked gasolines. This process is actually in use in this country on a commercial scale in three of the small refineries in the Mid-Continent. In one plant it has been applied directly to the vapor of a Cross cracking unit, in the others to rerun stills operating respectively on cracked distillates from Dubbs and Fleming units.

Satisfactory Results Claimed for Vapor Phase Process

All of these plants claim satisfactory results and two are increasing their capacities. The action of the fuller's earth on the cracked vapors passed through it seems to be quite definite, in its effect on color, color stability and gumming tendencies of the finished gasoline. The effect on sulfur content, while favorable, is to date not as far reaching as desirable, but some experiments are being conducted which I am told give considerable promise of success. One of the advantages claimed is that cracked gasoline so treated has a greater anti-knock value because of the comparative non-elimination of the olefines, etc., in contra-distinction to results with sulfuric acid treating and redistillation.

It is my opinion after going carefully over the work done so far, both laboratory and commercial, that this process promises the best hope

of a solution of the problem which has been offered to date. If the larger commercial installations contracted for and in prospect bear out the experience had so far, and both the treating and redistillation problems are solved thereby, the vapor phase process will truly constitute a basic change in refining methods.

Others are at work on the problem, attacking it from the angle of a purely chemical solvent problem, but attempting to find some more selective and less destructive solvent than sulfuric acid. One line of investigation is with various copper compounds, another is directed to the chromium derivitives. Patents have been applied for in connection with both lines of research but today none of the methods suggested by either group of investigators seem to have given a satisfactory commercial result, and all of them include a number of extra steps which would have a serious effect on the cost of operation.

The Edeleanu Process

The use of sulfur dioxide in liquid form, the Edeleanu process, has been applied commercially, so far by at least two large refineries on the Pacific seaboard, to the treatment of kerosene fractions from the partly naphthenic crudes of the asphalt base type produced in California. Results are said to be especially good in eliminating difficulties encountered with those products by the use of either ordinary or fuming sulfuric acids, and in producing a long-time burning oil. The plants are expensive to install, however, and require a high type of engineering skill to build, operate and maintain. So far the practical utility of the Edeleanu process for the treatment of the more stable paraffin base products has not been demonstrated. Nor has any commercial progress been made in applying the SO_2 method to the treatment of lubricating fractions in this country so far as known to me.

The Use of Fine Fuller's Earth in Refining

One more element in connection with the refiner's treating and decolorizing problems, which is assuming an importance sufficient to deserve mention as a basic improvement, is the growth of the use of fine fuller's earth both in the refining of gasoline and lubricating oils. This in a way, is a reversion to an old process. In 1896 George M. Saybolt, of Saybolt viscosimeter fame, applied for a patent on a process for neutralizing and further decolorizing hydrocarbon oils by the use of fine fuller's earth. This method was in extensive use for many years, principally on gasoline and kerosene, and for a short while also on lubricating oils, in time being displaced by the percolation filtration process. The discovery of new types of fuller's earth west of the Rockies, too fine to use as percolating material, revived interest in this method.

It seemed to offer a solution for a number of difficulties experienced in refining lubricating oils made from asphaltic and naphthenic base crudes, and has also found application in both overhead neutrals and residual cylinder stocks produced from paraffin and semi-paraffin base crudes. There are quite a number of commercial-sized installations in various refineries throughout the United States.

Progress in Handling Amorphous Wax

No important changes in the separation and refining of crystalline wax have been made in recent years, but real progress has been accomplished in handling the so-called amorphous wax, more commonly referred to as petrolatum. The fundamental basis of all methods of separating petrolatum from the containing petroleum liquids is that liquid wax contracts in volume when solidified by cooling, and its specific gravity is raised.

Wax Separation Twenty-five Years Ago

To visualize the progress that has been made in this branch of refining, let me recount a little history. Up to 25 years ago, the only known method of wax separation was to fill a large storage tank with Pennsylvania crude oil in the fall of the year and depend upon the gradual, steady cooling during cold weather to bring about the solidification and settling of the higher melting point wax crystals into the lower half of the tank. The high-gasoline content of Pennsylvania crude oil acted as the diluent. If the weather was propitious a fairly good separation took place in from 45 to 90 days. An interlude of warm weather of sufficient duration to raise the temperature of the oil would frequently destroy the effects of a couple of months' settling by reversing the convection currents in the tank, and many times all progress would be effectually stopped by protracted zero weather, which cooled the crude oil so rapidly as to make it congeal.

Slow progress was made for many years along the lines of artificial cooling and separating the highly diluted lubricating oil-petrolatum mixtures. With each successive step the results were a little better and, of course, the refiner became independent of the weather as soon as he resorted to artificial refrigeration. Though the dream of using centrifugal force was conceived many years earlier, it was not until 1917 that the experts of the Sharples Specialty Co., looking for new fields to conquer, tackled this problem for the refiner. They perfected the machinery which now forms an important part of the equipment of a great number of the more progressive lubricating oil refineries, operating on waxbearing crude oils.

Other Refining Improvements

A number of worthwhile improvements in less important departments have contributed a considerable share to the general march of progress. For instance, storage evaporation losses are being materially decreased as new preventive or saving devices are installed. Low-pressure control vacuum and pressure relief valves on storage tanks; the balloon breather bag principle; the insulated roof sometimes combined with fully insulated sides; and lastly, the floating roof, which eliminates the vapor space, is claimed to remove fire hazard and to cut losses to the irreducible minimum.

Concentrating Recovered Sulfuric Acid to Normal Strength

Another illustration: Two processes have been developed in recent years, both far superior to the old lead pan and still method, for concentrating weak recovered sulfuric acid to normal strength. One employs as one of its most distinguishing features, the Cottrell precipitator; the other's most novel characteristic is that all of its operations are carried on under a vacuum of 26 to 28 in. of mercury, with consequent low temperature operation. Each of them represents a true economy to the industry and each was conceived outside the industry proper.

Viewed in retrospect the changes appear revolutionary and kaleidoscopic. The time-honored practices of previous years seem to have been built upon or altered or discarded with bewildering speed in deference to the ruthless march of progress. Truly we must look back over our path to appreciate the steepness of the hill we have climbed.

Comparative Value as Motor Fuel of Cracked Gasoline vs. Straight Run Gasoline*

By Arthur E. Wells, New York, N. Y.

[Summary of paper presented before the Petroleum Division of the American Institute of Mining and Metallurgical Engineers.]

In this paper are described various comparative tests made on cracked and straight run gasolines. The results of these tests may be briefly summarized as follows:

Samples of 54 to 55° Bé. gravity cracked gasolines, with end point of about 442° F., and with a somewhat larger proportion of the less volatile fractions than conformable to government specifications, were used in a single cylinder variable compression engine under the same conditions as were samples of straight run gasoline of 58 to 59° Bé. gravity, and of which one had a greater proportion of the more volatile fractions than called for in government specifications.

The cracked gasolines showed decidedly less tendency to knock, hence could be used at higher compression ratios and thereby developed from 10 to 20 per cent. more power than did the straight run gasolines. These figures may be taken as one measure of the superiority of the cracked gasolines under test with the straight run gasolines.

It is not claimed that all cracked gasolines are superior to all straight run gasolines to the extent indicated by the above tests; cracked gasolines vary as do the straight run products. Cross's work and that of others have indicated that certain cracked products may have a much greater superiority over straight run than that determined above.

From various sources and by different methods of testing data are being obtained to show the definite and generally quite considerable superiority, as an anti-detonating fuel of cracked gasolines over straight run gasolines.

It is now becoming generally recognized that by increasing the production of cracked gasoline motor fuel from those heavier petroleum fractions, which otherwise would be used for burning purposes, the petroleum industry is not only working in the interest of the conservation of the world's supply of petroleum, but is at the same time making available to the gasoline-consuming public a product distinctly superior to the straight run gasolines.

* Petroleum Development and Technology in 1925, 448. (A. I. M. E. Pamphlet No. 1570-G, issued with Mining and Metallurgy, April, 1926.)

Review of American Production*

By F. Julius Fohs,† New York, N. Y.

(New York Meeting, February, 1926)

[Summary of paper presented before the Petroleum Division of the American Institute of Mining and Metallurgical Engineers.]

A comparative balance sheet for 1924 and 1925 follows:

	1925 Barrels	1924 Barrels
Gross production	763,000,000	714,000,000
Imports of crude oil	62,005,000	77,776,000
Imports of refined oil	16,263,000	16,470,000
Total	841,268,000	808,246,000
Surplus supply	25,000,000	30,246,000
Indicated domestic demand	704,465,000	667,000,000
Exports crude oil	13,353,000	17,913,000
Exports of refined oil	98,450,000	93,087,000
Total	816,268,000	778,000,000
Stocks, Dec. 31 (crude 410,554,000)	538,000,000	513,000,000

These figures show an increase of 49,000,000 bbl. in production and little change in oil imported; 37,000,000 bbl. greater apparent domestic consumption and actually greater utility in consumption, due to increased gasoline production by cracking; and 1,000,000 bbl. more crude and refined oil exported in 1925 than 1924, representing a total apparent increased demand of 38,000,000 bbl.

The excess of demand over supply in 1925 was 25,000,000 bbl. The increase of stocks over those at the close of 1924 was 25,000,000 bbl. Of the stocks on hand at the close of 1925, about 82,000,000 bbl. in California were not refinable.

Imports from Mexico and South America

It would not be surprising if, by 1930, we should actually welcome the imports from Mexico, Venezuela, Colombia, Peru, Argentina and Trinidad. If these States produce 155,000,000 bbl. and use, perhaps, 30,000,-

* Petroleum Development and Technology in 1925, 547. (A. I. M. E. Pamphlet No. 1570-G, issued with Mining and Metallurgy, April, 1926.)

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000 bbl. (a high figure) it will leave 125,000,000 bbl. for export; but part of this will go, as previously, to Europe and elsewhere. The estimated decline in Mexican production for 1926, of 14,000,000 bbl., will just about offset the expected production increase in South America; hence there is apt to be no material increase of imports into the United States in 1926, and we shall, therefore, be obliged to make up the production deficit for 1926 from the stocks on hand—a salient thing for the industry, though increased cracking will also play a part.

American Production

Approximately 400,000,000 bbl., or 52 per cent., of the 1925 United States production came from 27 pools, reaching the peak of their production within the past three years, and eight of these 27 reached their peak in 1925. Production from these pools will possibly decline one-third in 1926, since not to exceed four of them have possibilities for extensions or increased deep-sand production.

Improved Production Methods

If improved production methods were effective in two-thirds of the recent major pools during 1925, in excess of 106,000,000 bbl., 25 per cent. of such major pool production and 14 per cent. of the total production of the country for 1925, may be roughly ascribed to such methods. Actually, however, such methods were also used in the case of new small pools, as well as in the resuscitation of wells in old pools; so that it is probably safe to say that one-fifth of the production for the year may be attributed to these methods. An offset to improved production methods, even in the face of better geologic investigations and drilling methods, was the large number of dry holes drilled.

Output by Districts

California held front rank in 1925 and bids fair to hold this rank in 1926 and possibly 1927. During 1928 Oklahoma-Kansas or California may hold first rank and it will probably be 1930 before Texas reaches this position, due to increased Gulf Coast salt dome production.

The great factors in maintaining production in the Oklahoma-Kansas fields are: (1) The production backlog of recent flush pools; (2) possible major-pool developments from the small number of recent discoveries, already mentioned; (3) the finding of new Wilcox pools; (4) new pools west and south of the present producing areas; (5) the possibilities of new pools in the Red bed and Cretaceous-covered areas further west in Kansas and Oklahoma.

With the large amount of new leasing and drilling activity centered in Texas at this time, an increase in production is to be expected, though the Mexia fault zone, which has offered the great flush production for several

years past—except for possibilities of its southwast extension—must be replaced by West-Central Texas and possibly Northeast Texas for big gusher production. The relative importance of Texas districts in 1925 was as follows: Mexia, North-Central Texas, Gulf Coast, Luling, Western, or Reagan; Panhandle, Southwest Texas; while in 1926, North-Central and Gulf Coast will vie with each other for first place, with the Mexia district third, and Western Texas, fourth.

Both Arkansas and Louisiana should yield further pools for the insistent wildcatter, but none of consequence is in sight.

Salt domes were discovered in the Gulf Coast fields in 1925, but their real significance will not make itself felt until well toward the close of 1926 and possibly not until 1927. Geophysical methods will undoubtedly aid in the location of other domes in 1926, but the full effect of such discoveries will not be felt by the industry before, possibly, 1929 or 1930.

The Rocky Mountain District declined 26 per cent. from 1924, and a 10 per cent. decline was expected in 1926 from 1925. The Mississippi Valley fields can hardly be depended upon to maintain for 1926 production equal to 1925, whereas the Appalachian fields will do well in 1926 to approximate their 1925 production. Alaskan production is negligible.

DISCUSSION

In discussion, J. E. Pogue, New York, N. Y., pointed out the difficulty of forecasting crude oil production many months ahead because of not knowing what the new discoveries would be. It all depended in 1926 upon whether or not "we have two or three major pools; that is the big unknown factor in our production . . . If this year we have the analogues of all the new developments of 1925, with the exception of a Deep Smackover, our production will certainly be less than it was last year."

Whereas C. C. Osborn, Ponca City, Okla, granted that production in the United States in 1926 might be less than in 1925, he could not agree with the author that there would be a reduction of 100,000,000 bbl. He was of the opinion that enough new fields were in sight to greatly cut down the net effect of the estimated decline. Mr. Osborn felt that the low point in production for the United States in 1926, may have been passed during the week of Jan. 30. He said, in part: "There are areas in the United States where oil pools may produce over 100,000 bbl. per day within a few months after discovery. Can pools of this type be foreseen and measured in a long-range estimate?"

Mr. Osborn expressed the belief that "with the data available and known methods of measuring the production of oil pools, an estimate within a degree of accuracy which will be helpful rather than dangerous is impracticable for more than 90 days in advance."

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(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the titles of papers in *italics*.)

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